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Froth flotation of Flin Flon ore. 1933.

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The undersigned hereby certify that they have read and recommend to the Committee on Graduate Studies for acceptance, a dissertation on the "Froth Flotation of Flin Flon Ore" submitted by John Frederick Kostash, B. Sc. in partial fulfilment of the requirements for the degree of Master of Science.

Professor

Professor

Professor

LABORATORY TESTS IN FROTH FLOTATION OF FLIN FLON ORE.

- by -

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A THESIS

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PREFACE AND ACKNOWLEDGEMENTS.

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C H A P T E R I.

INTRODUCTION

DEFINATION OF FLOTATION.

Weinig and Palmer (1) define flotation as follows:- "Finely divided ore is agitated in water containing bubbles of air or gas. A small amount of ore or more reagents may be present, but not necessarily. Under these conditions the small particles of the so-called floatable minerals show a tendency to attach themselves to the films of the bubbles, which then carry the particles to the surface of the water. A small amount of a so-called frothing oil is generally added for the purpose of increasing the stability of the bubbles, so that they can be removed, taking their load of mineral particles with them. The floatable minerals tend to absorb a film of air or other gas, rather than a film of water, where-as the reverse is true of the minerals that are difficult to float. The latter minerals, because of the absorbed film of water, tend to remain in suspension in the water or sink to the bottom of the vessel. Thus there can be affected a more or less complete separation of the two classes of minerals, assuming, of course, that the grinding has been sufficient to unlock all of the particles of different composition. The separation does not depend upon differences in specific gravity; in fact, the

sulphide minerals, which are heavier than the oxidized minerals, ordinarily are the easiest to float. A particle of mineral may be so large, however, that its weight will counteract its affinity for the film of the bubble."

FLOTATION REAGENTS.

Chemical reagents added to the pulp perform various functions. Those which lower the surface tension of the water and thus produce a stable froth are known as frothing agents. The more common frothing agents are,- pine oils and cresols ($CH_3.C_6H_4.CH$). Those which form a film coating on the mineral particles and thus cause the particles to attach themselves to the air bubbles are known as collecting agents. The more commonly used collecting agents are,- potassium ethyl xanthate ($C_2H_5.OCSSK$), thiocarbanilid ($C_6H_5.NH.CO:SH.C_6H_5$), oleic acid (($(C_{17}H_{33}O_2)_3C_3H_5$), creosotes and crude oils. Other functions of the flotation reagents are to produce certain changes upon the particles of ore so as to increase or decrease the tendency of the collecting agent to coat the particles. These are known as activating, depressing, conditioning and sulphidizing agents. The activating agents increase the floatability of the minerals; copper sulphate ($CuSO_4.5H_2O$) is the only one known. Sulphidizing agents form a sulphide film on the mineral surface; thus causing

the mineral to act as a sulphide, sodium sulphide (Na_2S) is generally used. Depressing agents are those which decrease the floatability of the minerals; some of the more important depressants are,- lime (CaO), cyanides (RCN), sodium silicate ($Na_2Si_4O_8$), zinc sulphate ($ZnSO_4 \cdot 7H_2O$), and sodium bichromate ($Na_2Cr_2O_72H_2O$). Conditioning agents give alkalinity or acidity to the pulp solution. For acid pulps sulphuric (H_2SO_4) or hydrochloric (HCl) acids; and for alkaline or neutral pulps, lime, soda ash (Na_2CO_3) and cement are generally used.

TYPES OF FLOTATION OPERATIONS.

There are three types of froth flotation processes. (1)The agitation type, which depends upon the local supersaturation of the water of a pulp with air by the mechanical action of a swiftly revolving beater, and the simultaneous preferential precipitation of the air in the form of bubbles on the metallic mineral particles. (2) The pneumatic type in which the air is introduced through a porous medium at the bottom of the flotation cell. (3) The sub-aeration process in which the air is introduced into the pulp below the impeller by reason of the centrifugal force induced by an impeller revolving in the pulp.

FACTORS INFLUENCING FLOTATION.

From the defination given it is evident that there are many factors which influence flotation. This is especially true

for selective separation of the minerals in complex ores such as the Flin Flon ore. For concentration and separation of the minerals in the Flin Flon ore to give a copper concentrate, a zinc concentrate, probably an iron concentrate and a fairly clean tailing; some of the factors which have to be taken into consideration are,- mineral associations, size of mineral particles, chemical composition of minerals and its effect on floatability, pulp water, contaminants, temperature and time of treatment.

Mineral Associations. The manner in which the various minerals are associated with each other will influence the separation and concentration of the minerals into various products. Ores consisting of hard and soft minerals, or of brittle and tough minerals, fracture, in the coarse of grinding at the junction of the dissimilar grains. The softness and interstitial character of some sulphides also favor their liberation and their becoming ground to such a fineness as to present enhanced flotative properties. It can also be said that structural features of minerals belonging to different generations are different, the oldest minerals are fractured and cemented by younger minerals. Gaudin (2) states,- "In the ores of this type the matrix is generally softer and disaggregation takes place readily at the boundary between the grains because of the sudden change in mechanical properties at that boundary. These properties result in the production of comparatively coarse ore particles derived

from the older fractured mineral and of finer ore particles derived from the vein filling. Pyrite frequently occurs as a primary mineral with chalcopyrite and sphalerite fillings." This is true of Flin Flon ore where pyrite is the primary mineral and secondary chalcopyrite and zinc blende are the vein fillings.

Size of Mineral Particles. The texture of an ore is an important factor in determining the size to which the ore should be ground in order to liberate its minerals into particles consisting of one mineral only. Coarse grained ores do not require fine grinding to liberate the mineral particles but fine-grained ores require finer degree of grinding. In some cases where extremely fine grinding is necessary to liberate the mineral particles, there occurs a very poor recovery of metallic minerals in the fine sizes of flotation pulps which is not only due to the greater abundance of oxidized minerals in those sizes but also to non-flotation of particles that apparently are perfectly suited to float except for their fineness. Gaudin (2) who carried out investigations on the floatability of various particle sizes, states,- "The only interpretation that can be placed on the results is that there is a gradual decrease in readiness to adhere to air bubbles from coarse to fine particles, and that this tendency is manifested by an acutely perceptible reduction in recovery on proceeding from medium-fine (300 to 1000 mesh) particles to very fine particles (under 6000 mesh)".

Chemical Composition of Minerals. In the separation and concentration of ores by flotation, the individual chemical properties of the surface of the ground ore particles, rather than the chemical composition of the ore as a whole, are the controlling factors that cause the particles to either float or sink, and determine the value of the ore. The physico-chemical reaction in flotation is between the surfaces of the mineral particles and the flotation reagents, for example, zinc blende is easily floated when given a xanthate coating, but if zinc sulphate is added to the pulp. zinc blende will take a zinc sulphate coating and will be less easily floatable even with most active collectors. This same zinc-sulphate-coated particle can be made to float readily by coating it with copper sulphate. Surface coating of mineral particles may be accomplished and controlled by the mill operator by adding certain chemical reagents. But surface coating of mineral particles may, sometimes, be due to secondary enrichment or to oxidation. Gaudin (2) has this to say about secondary enrichment,- "Although similar in some respects to the later stages of primary mineralization, secondary enrichment differs from them in that the sulphide grains have reacted with the metalliferous solution, which may or may not be the case with the later stages of primary mineralization. In other words, secondary enrichment occurs as coatings upon the surfaces of the primary sulphide grains and

may contain the same metal or a different metal from the primary grains." From the point of view of flotation the coating of mineral particles by secondary enrichment is a very important factor and in some cases an undesirable one. For example, a pyrite grain may be coated by chalcopyrite by secondary enrichment and when floating for copper, this copper coated pyrite will be floated. In such a case it is necessary to grind to such an extent as to tear the chalcopyrite coatings from the pyrite in order to separate the sulphide copper from the barren pyrite.

Tarnished surfaces of sulphides have flotative properties more like those of oxides than like those of untarnished sulphide surfaces, and are less easily floatable. Tarnishing or surface oxidation of ore particles may take place in the ground before mining operations begin; it may also take place during the mining operations themselves which allow mine water and air to act upon the ore. Oxidation cannot be prevented, but during mining and dressing operations, it can be decreased by decreasing the length of time from the time the ore has been broken in the stope until it has been concentrated.

It may be said that chemical composition of the minerals has certain effects on the floatability. A sulphide mineral particle is more easily floated than a non-sulphide particle, but even among the sulphides there is a slight difference in floatability. For instance, with the use of a frother only, the

order of floatability is as follows:- copper, lead, zinc, nickel, cobalt, and iron. But if a collector is added the order of floatability is the reverse of the above series. The order of floatability with the collector roughly corresponds with the order of oxidation of the sulphide minerals. By using an oxidizing agent, it is possible to float the sulphides in the following order:- copper, lead, zinc, and iron. Hence, it is the chemical composition of the minerals which will determine the order of separation of the minerals in the ore.

Sometimes, soluble portions of the ore exerts a great influence upon the recovery of metal and upon the grade of the concentrate obtained. This is true in the case of the Flin Flon ore which has a little soluble copper salt produced after grinding, and which tends to promote the flotation of the zinc in the recovery of copper. The influence exerted by the soluble portions of the ore in most cases can be controlled.

Pulp Water. The purpose of water in flotation is to maintain the ore in a partially suspended, dispersed condition so that the particles may come into intimate contact with the reagents and the bubbles; to thoroughly wet the gangue; and to dissolve, or take into suspension, the various reagents that are used. The ratio of water to ore varies from four-to-one to one-to-one, depending on the grade of ore. With a very high dilution of pulp and low grade ore, each bubble in the froth carries fewer mineral

particles, and loses stability, since it is the mineral particles adhering to the bubbles which give the stability. Hence, the amount of water used has an important bearing on flotation; and when a satisfactory pulp dilution has been obtained in any given case, every effort should be made to keep this condition constant. Any variation in the dilution affects the reagents concentration, and also the action of the different physical and chemical forces.

Contaminants. The purity of the water used in flotation should be taken into consideration. A fresh piece of sulphide mineral may be floated readily in distilled water with proper reagents added. The same piece of mineral in a given ore pulp or in a given water may not float at all when the same reagents are used. Sometimes, foreign matter or contaminant, by acting as a depressant for a certain mineral may be helpful in a selective flotation. The foreign matter in water will vary from time to time. This factor cannot be eliminated nor very well controlled.

A large proportion of the slimes in the ore pulp, derived from what Gaudin (2) assumes to be "the microcrystalline alteration products of primary minerals by kaolinization, chloritization and sericitization" act as contaminants.

Temperature. Temperature of the pulp has an effect on flotation. Warm solutions require a smaller amount of reagents and give speed and intensity to the physical and chemical reactions. Most minerals are less easily floatable in cold water, especially zinc.

Time of Treatment. All chemical reactions require appreciable and varied length of time for their completion. Lime requires ample time for the chemical reactions to take place and is generally added to the ball mill, while xanthate, reacting very rapidly, is added just before the froth is being taken off. This time interval is known as conditioning and proper time to be given for conditioning can only be known through experience. The period will vary according to the minerals present in the ore and the reagents used.

REAGENTS FOR COPPER - ZINC - IRON SEPARATION .

The reagents to be used to give the best results for a particular ore can only be found out by experiment. For copper-zinc-iron separation, lime (CaO) or soda ash (Na_2CO_3) could be used for conditioning and to give the desired alkalinity. In the case of soda ash, it would be necessary to use a depressing agent for iron if iron occurs as pyrite, as sodium carbonate has no depressing effect, whereas lime has a depressing effect on pyrite.

Pine oils and cresols are used extensively to give a

froth. The froth given by pine oil is more voluminous than that given by cresol. The choice between the two depends on the type of froth desired. Cresylic acid (cresol) has a slight selective action for zinc blende.

Ethyl xanthates are the most generally used collectors. Xanthates have little selective action and if added in sufficient amount, they will float nearly anything. Thiocarbanilid is also used as a collecting agent, but due to its low solubility in water, it is seldom used directly, but ordinarily, is used as a solution of about one part of the salt in four parts of ortho-toluidin known as TT mixture. The ortho-toluidin has little or no value as a flotation reagent, and is used simply as a solvent. Due to the high cost of this solvent, thiocarbanilid is not used to as great an extent as the xanthates.

The only activating reagent for zinc is copper sulphate ($CuSO_4 \cdot 5H_2O$). Zinc sulphate ($ZnSO_4 \cdot 7H_2O$) is a satisfactory depressing reagent for the zinc and iron. Sodium cyanide is an important reagent in flotation because of its great effectiveness in preventing the floating of zinc and iron minerals, but due to its high cost, it is used only where zinc sulphate cannot be used.

It has been found by the Denver Laboratories of the Complex Ores Recovery Company, Denver, Colorado (3), when carry-

ing out flotation tests on the Flin Flon ore, that sodium chloride ($NaCl$) had a depressing effect on the talc which floated with the copper minerals, thus lowering the copper concentrates.

The amounts of the various flotation reagents to be used depends on the nature of the ore and definite values can only be obtained from actual practice.

CHAPTER 2.

FLIN FLON ORE BODY

DISCOVERY.

Almost the first prospecting done in the region was in 1896, when Mr. Loucks (4) made an expedition to Reed Lake, where he staked a claim for gold. It was not until 1908, however, that any interest was taken in the region. About this time active prospecting was begun. In 1913, the finding of gold-bearing quartz veins on Amisk Lake caused a great deal of activity in the region. In the summer of 1915, the Flin Flon ore body was discoveredby the Creighton-Mosher party, and in the autumn of the same year the Mandy lens was located by Reynolds and Jackson. Since then the mineral belt north of the Pas, Manitoba, has attracted considerable attention.

LOCATION.

The Flin Flon ore-body (5) is situated at the south-east end of Flin Flon Lake. The ore-body lies in township 67, ranges 29 and 30, west of the first meridian, five miles east of the second meridian west, The latitude is approximately 54° 45' north and longitude approximately 101° 55 ' west,

lying in a right randant angle of the Interprovincial boundary and lies partially in the province of Manitoba and partially in the province of Saskatchewan, being cut in two places by the boundary line.

DEVELOPMENT.

After the discovery of the Flin Flon ore-body in 1915, some trenching was done that summer to obtain an idea of the size of the deposit. In the autumn of that year, claims were recorded. Diamond drilling was commenced in March, 1916, by Hayden-Stone (6) interests of New York and Boston, and in four months 6000 feet of drilling were done. Failing to come to an agreement with the owners, the work was suspended. In the spring of 1917, the Fasken interests of Toronto agreed with the owners to continue drilling, carrying on the work until July, 1918. In all, 44 holes were drilled representing a total of 25,664 feet.

In March, 1920, an option was taken on the property by the Mining Corporation of Canada jointly with the New York interests. During that year two shafts were put down and some cross-cutting and drifting done on the ore-body.

In November, 1925, the Mining Corporation interested the Minerals Separation Company, representing the Whitney interests

of New York. A series of laboratory experiments was carried out by the Denver Laboratories of Complex Ores Recoveries Company (7) and a flotation process was developed for the concentration of the chalcopyrite and zinc blende. In order to test the process under the conditions existing at the property, an experimental pilot mill, capable of handling 25 tons of ore per day, was erected on the property in 1927. The mill operated on fresh ore from underground mining operations until February, 1928, It was decided to develop and work the deposit and the Hudson Bay Mining and Smelting Company was formed, with the Whitney people holding a fifty per cent interest.

The suceess of the project required adequate transportation in the form of rail connection with the Hudson Bay Railway, a distance of 87 miles.

As soon as the steel reached Flin Flon, construction of a 3000-ton concentrating mill at Flin Flon and a power plant at Island Falls was begun. In June 10th, 1930, the transmission line was testedand current delivered to the main transformers at Flin Flon. Immediately, after the power was available, the mine, crushing plant and the concentrator equipment was placed in operation, but actual milling, concentrating and smelting operations did not commence till November, 19th, 1930.

From the date of full operation up to the end of 1931, 595 cars of blister copper and 430 cars of zinc were shipped. Production for 1932 should be close to thirty-seven million pounds of copper, and forty-five million pounds of zinc, as well as, two million dollars in gold and two hundred thousand dollars in silver, (8).

GENERAL GEOLOGY.

The geological features of the Flin Flon ore-body, is described in a report by R. C. Wallace (9), as follows:- "The ore-body lies in amygdaloidal greenstones which are to be referred to the earliest volcanic flows in this district. They have been named by E. L. Bruce (10)(who has mapped this area in detail for the Geological Survey), the Amisk volcanics, owing to their prevalence in the vicinity of Amisk (Beaver) Lake. Associated with the greenstones, and probably somewhat later in age, although earlier than the granite intrusions, are quartz porphyries which are parallel in strike to the greenstones (326°) and which do not here display clear-cut intrusive relationships into the greenstones. There are also lamprophyric dykes which are apparently earlier than the intrusions of granite, probably earlier than the quartz porphyry flows, though later than the amygdaloidal

greenstones. The later granite, which is found one mile southwest of the Flin Flon ore-body, and to which may be referred the granite porphyry intrusion immediately north of the ore-body, has been named by Bruce the Kaminis granite. This granite is intrusive, not only into the volcanics referred to above, but also into sediments that overlie the volcanics, but which are not exposed in the immediate vicinity of the Flin Flon property."

Shearing took place, probably during the earlier stages of the eruption of the Kaminis granite. During the process of shearing, the quartz porphyry offered greater resistance to shearing forces than did the amygdaloidal greenstone. In all probability, the zones that suffered intensive shearing represent a volcanic tuff interbanded with the more massive lava flows. In the process of replacements by sulphides, the more sheared zones have suffered much greater change than have the more massive members.

The Flin Flon deposit are to be referred to the Kaminis granite as ore deposit representing deposition at higher temperatures.

SHAPE AND EXTENT OF THE ORE-BODY.

The ore-body, which skirts the south-east shore

of the Flin Flon Lake, strikes north 30° west, and dips from 60 to 70 degrees to the north east. The diamond drill boring records show that, at the south end of the deposit, it pitches at a low angle to the south. In one place, quartz porphyry dyke forms the hanging wall of the ore-body. 'Horses' of unmineralized greenstone separate the ore-body into more or less independent lenses (Figure 1 and 2. Pages 19 and 20).

The total length of the ore-body, on the surface is 2,593 feet; at a depth of 900 feet, it has a length of over 1000 feet. It has greatest width at 400 feet depth; and at 900 feet, it narrows to 35 feet. It is estimated that there are 16,000,000 tons of ore available. The ore-body is most compact at the north end and shows a tendency to intermingle with inclusions of considerable widths of country rock towards the south end and at depth.

MINERALOGY.

Macroscopic. Macroscopic examination of specimens show the principal metallic minerals are pyrite, zinc blende and chalcopyrite with small amounts of gold and silver. The zinc blende is of two varieties, sphalerite and marmatite. Some galina occurs where zinc blende is abundant. Arsenopyrite occurs sparingly as clear cut crystals enclosed in pyrite.

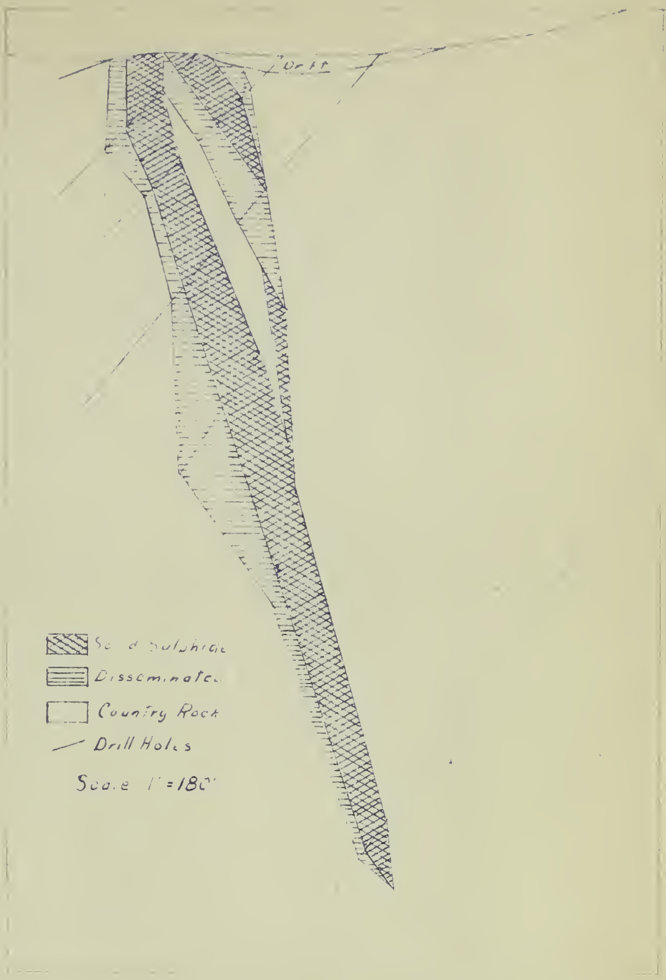


Fig 1 Section at north end of ore-body as determined by diamond drilling.

after R C Wallace

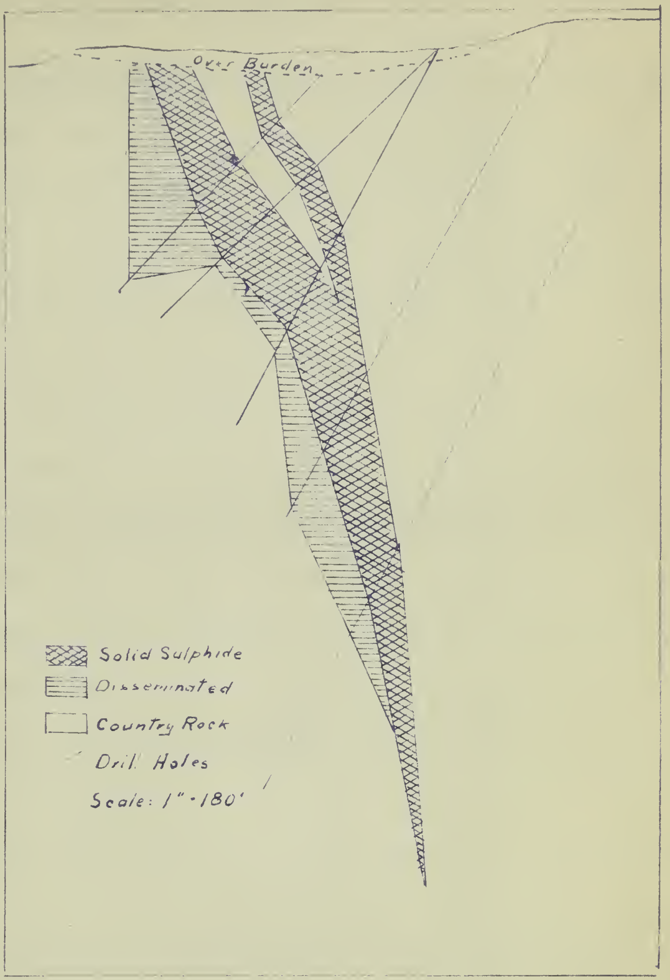


Fig. 2. Section at south end of ore-body as determined by diamond drilling.

after R. C. Wallace

There is no evidence of free gold or silver in the handspecimen, but assays indicate that these occur both with the sulphides and with the gangue minerals.

Microscopic. Microscopic examination of polished sections with reflected light, show the sulphides in a hypidiomorphic to allotromorphic structure indicating a progressive crystallization (Plate 1.). Pyrite in contact with gangue minerals as frequently idiomorphic (Plates 2 and 3.), indicating prior deposition. Idiomorphic crystals of pyrite indicate that it was the first sulphide to form (Plate 2.). Where present, arsenopyrite is intergranular with pyrite (Plate 4.). The relationship between sphalerite and marmatite is not clearly defined but there is some evidence that sphalerite surrounds the richer iron bearing marmatite (Plate 3.). This would confirm the pyrite-zinc blende relationship. Presumably the sulphide solutions were differentially impoverished in iron during the process of deposition. The order of deposition of the sulphide would then appear to be pyrite-marmatite-sphalerite. The relationships of the other sulphides, chalcopyrite, galena and arsenopyrite was probably prior to zinc blende (Plate 4.) and may have been contemporaneous with pyrite.

Examinations of thin sections of the ore specimens have shown the gangue minerals to be hornblende, actinolite,

chlorite, sericite, biotite, quartz and calcite (Plate 5.) J. F. Wright (11) carried out microscopic examination of rocks occuring in the Flin Flon area. He includes, besides the minerals mentioned above, the following,- epidote, feldspar, garnet, and others in lesser quantities.

Microscopic examinations of the ore specimens have shown that the sulphides penetrate the crystals of hornblende and other silicates and it is clearly seen that the sulphides were introduced later than the silicates. Some of the quartz has been introduced during the sulphide deposition as it is closely associated with the sulphide minerals and bears no relation to the gangue minerals which constitute the schistose rock (Plates 6 and 7.). Alteration of the feldspars, hornblende, actinolite and biotite has yielded chlorite, sericite, and talc in considerable quantities. These minerals give a banding appearance to the rock ~~and~~ polarized light between crossed nicols, shows that the minerals along the banding have a common extinction.

There is a difference of opinion as to the alteration of the silicate minerals during deposition of the sulphides. Alcock (4) suggests that the silicate minerals were altered by the hot solutions which brought the ore. The wall rock near the sulphide zone contains much sericite; and irregular

masses of talc have been found in chlorite schist and in the sericite schist of the footwall. J. F. Wright (11) draws the following conclusion:- "An interesting feature of specimens studied microscopically, is the small amount of alteration of the gangue minerals during the deposition of the sulphide. About fifty specimens of sulphide-bearing rock have been studied microscopically and these exhibit no evidence of widespread silicification or chloritization of the wall-rocks, which is a characteristic feature of some deposits of similar type in other parts of the world. Fresh silicates are included within the sulphides and the minerals adjoining veinlets of sulphides are unaltered. In some specimens biotite or hornblende is slightly bleached and some feldspar is altered to saussurite in small areas. ----- Chlorite and sericite are abundant in some sulphide bodies, but these minerals are also characteristic minerals of the metamorphosed rocks outside the deposit. Where pyrite crystals are developed these cut across foils of chlorite; bits of chlorite, sericite and epidote occur within the massive sulphides. Although in some specimens it is difficult to prove that the secondary silicates resulted entirely from metamorphism along shear zones before the sulphide bodies formed, nevertheless the available evidence suggests strongly that this is the important source of these minerals rather than alteration associated

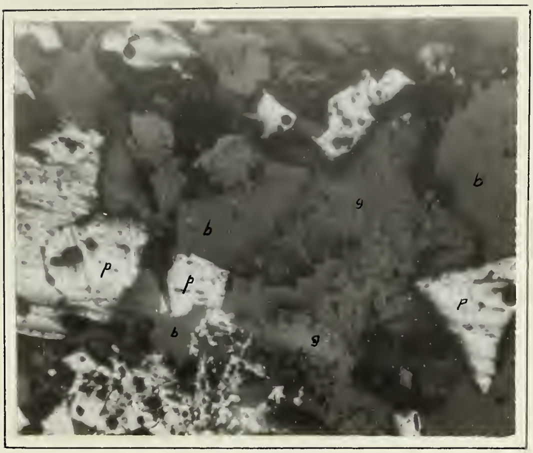


PLATE 1. POLISHED SECTION. 100X.

p. pyrite; b. zinc blende; g. gangue mineral.

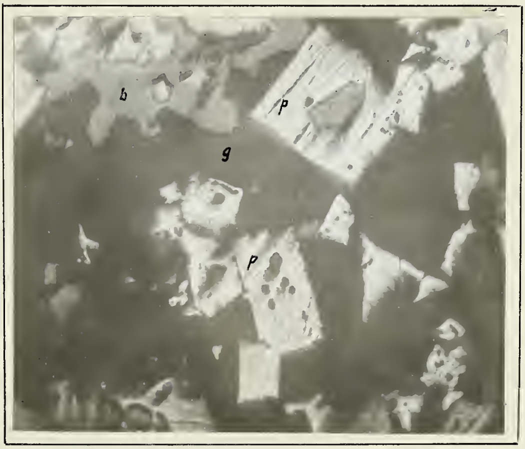


PLATE 2 POLISHED SECTION 100X

p. pyrite; b. zinc blende; g. gangue mineral.

PLATE 3. POLISHED SECTION 100X
p. pyrite; m. marmatite; s, sphalerite;
g. gangue mineral.

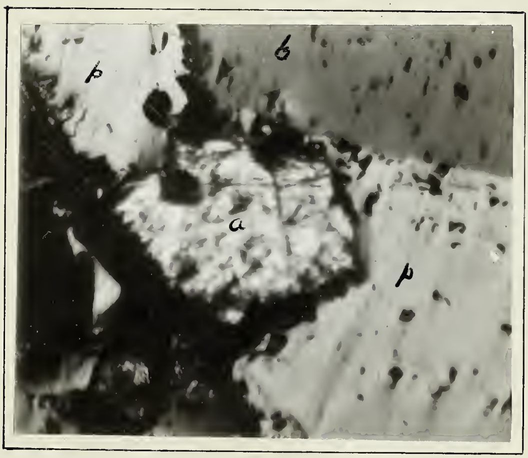


PLATE. 4. POLISHED SECTION 100X
a, arsenopyrite; p. pyrite; b, zinc blende.

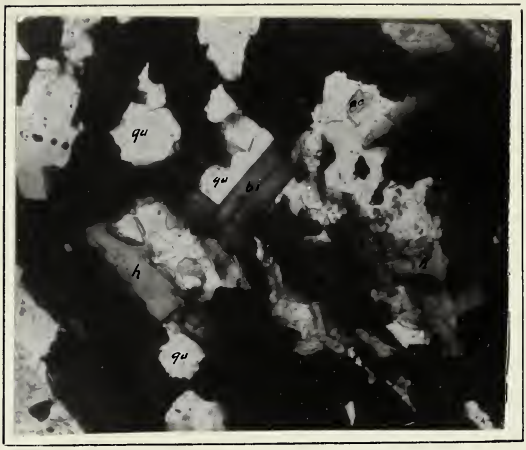


PLATE 5. THIN SECTION 100X
qu, quartz; bi, biotite; h, hornblende;
ac, actinolite.

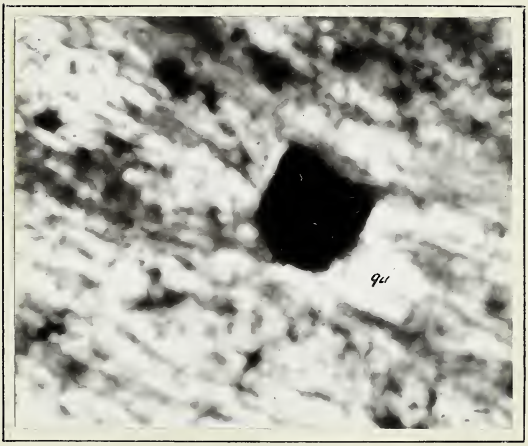


PLATE 6. THIN SECTION 100X
qu, quartz; black mineral is pyrite.

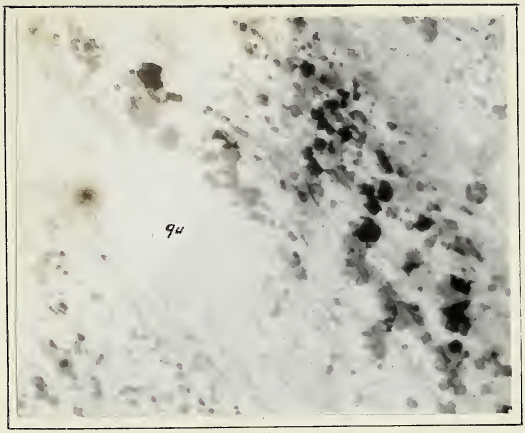


PLATE 7. THIN SECTION. PARALLEL LIGHT
qu, quartz; black mineral is pyrite.

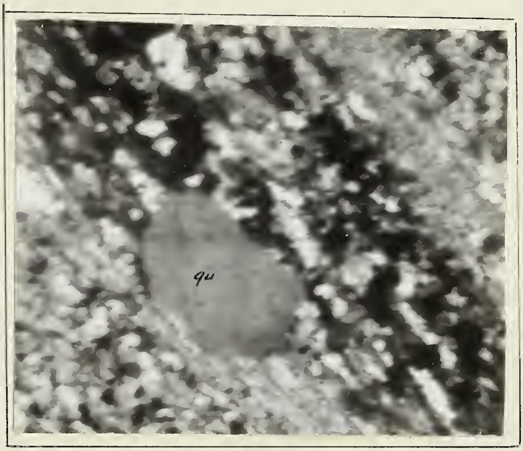


PLATE 8. THIN SECTION. NICOLS CROSSED
(showing common extinction); qu, quartz.

with mineralization."

TYPES OF ORE.

The ore consists of two fairly distinct types; solid sulphide and disseminated ores. The solid sulphide variety consists of very fine grained pyrite, containing zinc blende, chalcopyrite with occasional fragments of schist, quartz and calcite (Plate 2.).

In places the ore is banded where the zinc blende and the chalcopyrite form narrow bands in the pyrite.

The disseminated ore consists of schistose rock impregnated with sulphides (Plate 6.)

The solid sulphides occur in the centre and towards the hanging wall of the ore-body (Figures Nos. 1 and 2. Pages 19 and 20), and are, in places, in direct contact with the hanging wall; whereas the disseminated ore is largely confined to a zone along the footwall. Although it is sometimes found on the hanging wall side in the upper part of the deposit. The hanging disseminated ore is found to carry smaller percentage of copper and greater portion of gold and silver than the disseminated ore on the footwall. The zinc blende is more abundant on the hanging wall side. Contacts between the solid sulphides and the 'horses' of unmineralized rocks are quite sharp; also the boundaries be-

tween the two types of ore are fairly distinct, though in places a graduation between the two varieties is found.

ORIGIN.

The deposit is a replacement along the shear zone. The Kaminis granite and its offshoots seem to be the parent body from which the mineralizing solutions have come. Elsewhere in the area the sulphide bodies frequently occur in close association with quartz veins carrying gold. It is evident that the gold-bearing veins are the closing phases of the granite intrusions and it is probable that the sulphides were derived from the same source.

It seems that there was an earlier introduction ofpyrite which replaced and impregnated the crushed and sheared rock. Later movements fractured and granulated the pyrite, and the chalcopyrite and the zinc blende were introduced. The segregation of these minerals into individual zones may have been by selective precipitation. Since the country rock was schistose even at the time of the first mineralization, its replacement by pyrite would take place most easily along the planes of schistosity.

EFFECT OF MINERAL COMPOSITION ON FLOTATION.

The metallic minerals are sulphides of iron, copper

and zinc. Being sulphides, the ore should not present any difficulties in separation and concentration. As there were two periods of mineral deposition in the Flin Flon ore-body, there is a possibility that the copper and the zinc minerals, of the later deposition, coat the pyrite mineral particles, and thus make that mineral act as the coating. For example, a pyrite grain may be coated by chalcopyrite and when floating for copper this copper coated pyrite will go to the copper concentrate. In such a case it would be necessary to grind to such an extent as to tear the coatings from the pyrite in order to separate the minerals.

The zinc minerals are not purely zinc sulphides, they vary in composition from sphalerite to the iron variety, marmatite. It is not probable that the iron type would present more difficulties in separation than the sphalerite, but it is reasonable to expect higher iron in the concentrates due to the iron in the zinc blende.

Gold and silver are present in both sulphide and gangue minerals. That with the sulphide is recoverable in flotation concentrations, but that in the gangue minerals, requires excessively fine grinding to be liberated.

Chlorite, sericite and talc always present some difficulties in flotation. These minerals are flaky and present a large surface for a small mass, and are readily floated. Thus

they tend to come up with the metallic mineral concentrates, decreasing the grade of those concentrates.

CHAPTER 3.

PREVIOUS WORK ON FLIN FLON ORE

Investigations on the concentration and separation of the minerals in the Flin Flon ore were carried out by the Ore Dessing and Metallurgical Laboratories, Department of Mines, Ottawa, the General Engineering Company, Salt Lake City, Utah, and the Denver Laboratories of the Complex Ores Recovery Company, Denver, Colorado.

INVESTIGATIONS BY THE ORE DRESSING AND METALLURGICAL LABORATORIES, OTTAWA, IN 1921.

Investigations by the Ore Dessing and Metallurgical Laboratories(12) consisted of a large number of flotation tests, using the laboratory type apparatus on the differential flotation of the minerals in the Flin Flon ore. The object was to determine if the minerals could be separated into products suitable for reduction, or metallurgical treatment into metallic state.

The shipment received by the Ore Dessing and Metallurgical Laboratories consisted of 229 pounds of drill cores, crushed to about one-half inch size, from the exploration work on the massive sulphide zone.

Possible Lines of Investigation. On account of the

nature of the Flin Flon sulphide ore, which consists of practically solid sulphides of iron, zinc and copper with the gangue minerals less than ten per cent, it presented a most intricate metallurgical problem to determine the most economical method of treatment. Three of the most promising lines of investigations decided upon were as follows:-

1. The concentration of the copper mineral by selective flotation and the separation by differential flotation of the chalcopyrite, zinc blende and pyrite, resulting in copper and zinc products suitable for subsequent reduction to the metals.
2. The hydro-metallurgical treatment of the ore, or of concentration products, by roasting, leaching and electrical deposition of the metals.
3. Pyritic smelting followed by reduction of the matte in converter to blister copper.

The third method was not tried out, as it would require a study of conditions at the metallurgical plants operating on a similar class of ore, and also, would require an installation of a small unit at the property to demonstrate the feasibility of this method of treatment.

Tests on the Concentration and Separation of the Minerals.

The results of the tests conducted by the Ore Dress-

ing and Metallurgical Laboratories and the General Engineering Company are summarized in Table No. 1.

Roasting and Leaching Tests.

Roast No. 1.

A sample of 100 grams of ore, 200 mesh, was spread over an area of 8" x 3" in an electric muffle furnace. The doors were kept closed during the roasting. A suction was applied to the rear door to draw off the excess SO_2 gas. Time of roast was 6½ hours. Temperature range 300° - 650°C. Rabbling was performed every half hour.

Leaching. Two gram samples were leached at a point below the boiling temperature for 1 hour. The results were as follows:-

Head sample Cu	2.35%		
Water soluble Cu	0.58%	..... extraction	24.7%
H_2SO_4 soluble Cu	2.19%	..... "	93.2%
Head sample Zn	6.75%		
Water soluble Zn	4.36%	..... extraction	64.6%
H_2SO_4 soluble Zn	6.12%	..... "	90.7%
Insoluble residue from 4% acid leach			65.0%
Heads SO_3 in sample			10.5%

Roast No. 2.

A sample of 100 grams of ore, 20 mesh, was roasted in an electric muffle furnace. General conditions were similar to Test No. 1. with the exception that the front door of the furnace was left open slightly (½") to allow a current of warm

air to pass over the material. Time of roast was 3 hours, temperature range, - 450° - 610°C. Rabbled every half hour.

Leaching. Similar to Test No. 1. The results were as follows:-

Head sample Cu	2.35%		
Water soluble Cu	0.36%	..... extraction	15.3%
4% H_2SO_4 soluble Cu	2.18%	..... "	..... 93.0%
Head sample Zn	6.77%		
Water soluble Zn	3.95%	..... extraction	58.34%
4% H_2SO_4 soluble Zn	5.50%	..... "	..... 81.24%

Summary and Conclusions.

Conclusions drawn by the Ore Dressing and Metallurgical Laboratories from the above tests were as follows:-

1. The microscopic examination of the flotation products of Flin Flon ore showed:- "that each of the three chief sulphide minerals of the ore - pyrite, chalcopyrite and sphalerite - occur for all purposes practically free from any other sulphide or gangue minerals. --- Much of the material already ranges well below 200 mesh in size, and it would hardly seem feasible to even consider finer grinding because of the prohibitive cost.
2. "The assays seem to indicate that the gold and silver occur more or less indifferently with either the chalcopyrite, the zinc blende, or the pyrite. Hence flotation merely separates the chief sulphide minerals of the ore and leaves

the gold and silver almost equally disseminated in each.

3. "The results of the flotation tests show no difficulty in the concentration of the chalcopyrite by preferential flotation after grinding to 200 mesh. A copper concentrate is obtained as high as 15 per cent copper with a recovery of 85 per cent of the copper values in the ore. There is no reason to doubt, as far as the copper is concerned, that these results could be obtained in practice --- The results show that a separation has been made of the --- sphalerite from the pyrite and gangue. A zinc concentrate, 40 per cent zinc, with a recovery of 50 per cent of the zinc values, has been obtained.

4. "The results of the roasting and leaching tests showed good extractions of the copper and zinc. These small tests were not conclusive with regard to feasibility of this process as applied to the ore. To obtain reliable data, tests would have to be conducted in a small plant over a period of time, so as to obtain conditions approaching practice (13)."

INVESTIGATIONS BY THE ORE DRESSING AND METALLURGICAL LABORATORIES, OTTAWA, IN 1922.

During 1922, testing was conducted by the Ore Dressing and Metallurgical Laboratories, Ottawa (14), on the concentration of the disseminated ore of the Flin Flon ore-body. Two shipments were received by the Laboratories, one of 130 pounds

and the other of 230 pounds.

The purpose of the experimental work was to determine if the ore was amenable to concentration, with the production of a concentrate in as coarse a form as possible, with high recovery of the copper values. Pyritic smelting of the sulphide ore having been decided as the better practice, the concentrate from the disseminated ore could be mixed with it and bring up the grade of the furnace charge.

<u>Lines of Investigation</u>. On an ore of this class (disseminated ore) the first line of investigation would be to determine if a copper concentrate could be produced by jigging, removing as much of the copper values as possible in a coarse form. This would be followed by tabling the flotation for the recovery of the remaining copper values, which are more intimately disseminated in the gangue, or floating off with the slimes.

If it were found that the copper values were not freed by crushing to a size suitable for jigging, the ore would be further reduced for table concentration, followed by regrinding and flotation from the table concentration of finer particles.

If it were found that a table concentrate could not be obtained after crushing to a size suitable for table work, the next line of investigation would be to grind the ore to a fineness at which the copper minerals were free with the production of a copper concentrate by flotation.

Experimental Tests by Jigging.

Test by jigging made on 1/4 inch material showed that jigging was not practicable. The concentrate obtained was very low in grade, consisting of all the sulphides. The tailing contained a large amount of unfreed copper mineral.

Table Concentration of the Sized Material.

The object of this test was to determine if tabling could be used in conjunction with flotation.

7000 grams of ore, crushed to pass 20 mesh, was sized and each size was tabled separately. The results were as follows:-

-20+28 -- the separation was very poor, and no distinct line of sulphides was obtained.

-28+35 -- the separation was no better.

-35+48 -- the separation of the sulphides showed a marked improvement on this size. The proportion of middling obtained was smaller. The middling was re-run over the table, but no separation could be obtained showing the product was a true middling. The middling was added to the tailing.

-48+65 -- the separation of this size was good. Both the tailing and concentrate looked clean. Very little middling was obtained.

-65+100 -- the separation was better than in previous size.

-100+150 - the separation looked very good, but considerable float material went with the tailing. The tailing was re-run giving a distinct copper sulphide line which was practically pure chalcopyrite. The loss of copper by flotation in tabling this size would be appreciable.

-150+200 - the separation on the table was excellent. The sulphides separated into two lines. The sulphide band nearest the gangue line was distinctly higher in copper than the band of sulphides higher up on the table. The tailing looked very clean and free from sulphides, but the analysis showed that the tailing contained 1.90 per cent copper, a low recovery.

Flotation Tests.

The next series of tests made were by flotation. Coal tar oils and fuel oils were used in the first few tests. These were unsuccessful; nothing but a voluminous froth was obtained, consisting of fine flaky gangue slimes. What actually happened when flotation of the copper was attempted in the presence of the talcy gangue, was that the material being in the form of flat thin flakes, presented a large surface area, and completely surrounded

and adhered to the bubbles thus excluding the chalcopyrite from the interfacial boundaries. Various reagents were tried to hold down these slimes and coagulate them, but with no success. It was found that lime had the greatest effect on the coagulation and settlement of the talcy material. The first test made with thio-fizzan in an alkaline pulp with lime gave the results as indicated in test No. 2. (see table No. 1.) The results were fairly good.

It was apparent that the talc slime interfered with the flotation. Hence the following series of gravity tests were run to determine if the gangue could be separated from the rest of the ore.

Gravity Test.

The ore was dry crushed to 20 mesh and tabled without sizing on a small laboratory wilfley table. The results of Test No. 3. were, as given in Table No. 2.

The three sand products from the table were mixed together and a sample weighing 1000 grams was cut out for a flotation test, conducted similar to Test No. 2, the results of which are tabulated in Table No. 1. Test No. 3.

The recapitulation of the complete Test No. 3. shows that a total recovery of only 75 per cent of the copper was obtained in the concentrates assaying 11.5 per cent copper and re-

presenting 16.5 per cent by weight of the original ore.

Tests Nos. 7 and 8 were made to determine if the talcy material could be eliminated in the tailing without excessive loss of copper. It was thought that perhaps wet crushing would free this talcy gangue material better than dry crushing which had been tried in Test No. 3. A 1000 gram sample of ore was prepared by dry crushing to pass 20 mesh, then placed in a ball mill for five-minute period for wet crushing. The crushed product was, then, tabled. The results are tabulated in Table No. 2.

A series of tests, Nos. 11, 12 and 13 were made using hydraulic classification after coarse crushing to eliminate the slimes interfering with flotation. 1000 grams of ore - 1/4" were crushed in wet ball mill for 5 minutes and the sands and slimes of the resulting product were separated in a Richard classifier. The sands were then reground to 100 mesh for flotation and the separated slimes tabled. The results are recorded in Tables Nos. 1 and 2.

<u>Flotation Tests</u>.

The next step tried, in the concentration of the Flin Flon disseminated ore, was to see if better recovery could be obtained by the removal of talc by preliminary flotation prior to flotation of copper than by gravity concentration and froth

flotation.

The procedure was as follows:- the ore was crushed by dry crushing in rolls to 20 mesh. A 1000 gram lot was placed in a small ball mill. If lime was used it was added to the ball mill. The pulp, of ratio of one to one, solid to water, was ground to the desired size product. The flotation tests were conducted in a Ruth laboratory machine. The results are given in Table No. 1. Tests Nos. 15, 21b and 22b.

It was observed that for the same reagents and the same amounts of reagents the recovery and grade were not the same. It was thought that the size of crushing had an important effect on the results, and a series of tests, Nos. 26, 27, 28, 29, 31 and 33, (Table No. 1.) were made to give results of flotation on different size products, and with various reagents.

A straight flotation method was tried with middling returned to the circuit (Test No. 36). The results are shown in Table No. 3, Test No. 36.

The results show that, returning the middling to be mixed with the feed to the rougher cell is detrimental. The middling builds up in the circuit and loads the circuit up with froth, finally reducing the grade of the concentrate, and increasing the loss of copper in the tailing.

Tests Nos. 44, to 54 (Table No. 1.) were made to determine the froth removing properties of the various Fumols.

Tests of General Engineering Company.

A small sample of ore was sent to the General Engineering Company, to check the results of the Mines Branch. Tests, by the General Engineering Company, were made in a Callow type pneumatic machine. The results are recorded in Table No. 1.

Results of Tests.

The gravity concentration tests on the Flin Flon disseminated ore showed ~~high~~ low grade copper concentrate. Flotation tests showed that separation and concentration by froth flotation process was possible; and that a preliminary removal of talcy mineral was necessary in order to yield high grade copper concentrates. As high as 85 per cent recoveries of copper were obtained, running as high as 18 per cent copper. The copper loss in the talc froth running from about 5 to 10 per cent and in the tailings about 7 per cent.

The recovery in the Ruth machine ~~is~~ was higher than in Callow when using straight flotation. A 0.25 per cent copper tailing was constantly obtained in the Ruth against a 0.50 per cent or higher tailing in the Callow. This is due to the fact

that in the Callow, the middling is returned continuously to the rougher cell (Tests Nos. 31 and 38). When the talc was removed by preliminary flotation approximately the same results were obtained in both machines (Tests Nos. 15 and 37b).

The work of these tests showed that there were four possible flow sheets for the treatment of the Flin Flon ore, as follows:-

1. Straight flotation, but middling from the cleaner cell should be, either treated in a separate circuit or tabled to remove the talc before being returned to the flotation circuit. Flow sheet No. 1. Figure 3.
2. Straight flotation with middling returned to circuit. Flow sheet No. 2. Figure 4.
3. Removal of the talc by preliminary flotation with a reagent which will float the talc without having a collecting action on the copper, followed by flotation of the copper. Flow sheet No. 3. Figure 5.
4. Removal of the bulk of the talc by hydraulic classification at different points in the grinding circuit and flotation of the modified pulp as in (1).

After considerable further investigations, as reported in what follows, method No. 3. was finally accepted as the best

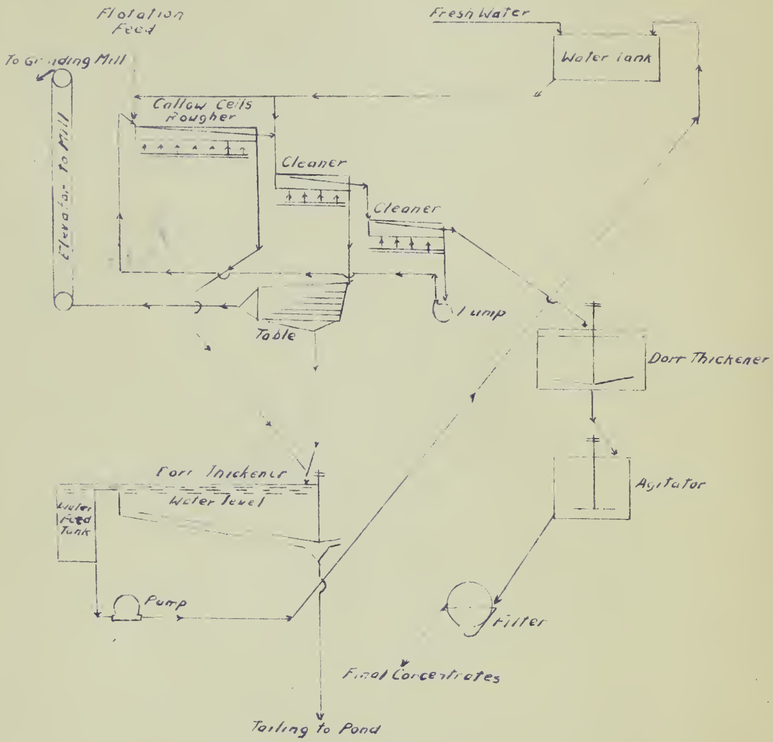


FIG. 3 FLOW SHEET NO. I.

after C.S Parsons.

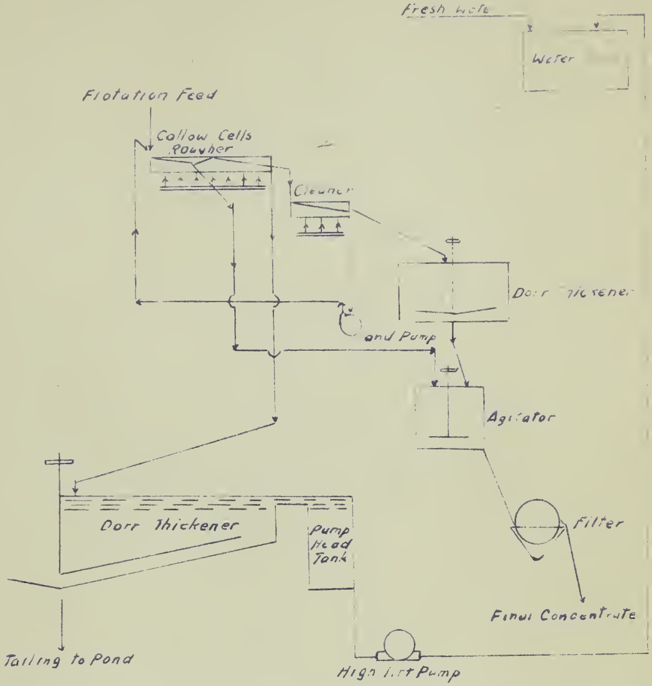


FIG. 4 FLOW SHEET NO. 2.
after C. S. Parsons

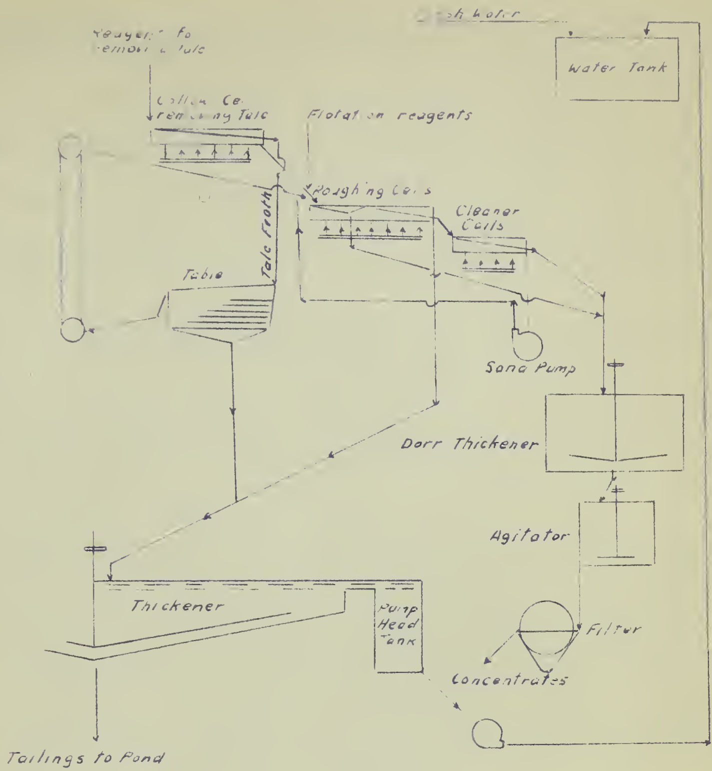


FIG. 5. FLOW SHEET NO. 3.

after C.S. Parsons.

method for treatment of the Flin Flon ore.

INVESTIGATIONS BY THE DENVER LABORATORIES, COLORADO, IN 1925.

Experimental work by the Denver Laboratories (15) consisted of concentration and separation of the base metals. Most of the work was made on the massive sulphide ore, although a few flotation tests were carried out on the disseminated ore.

Lines of Investigation. The nature of the massive sulphide ore suggested the following lines of investigation:-

1. Direct leaching of the raw ore to recover the minerals of chief commercial values.
2. Hydro-metallurgical treatment of the ore by roasting, leaching and electrical deposition of the metals.
3. Concentration of the copper and the zinc minerals by differential flotation resulting in products suitable for subsequent reduction to the metals.

Direct Leaching.

An attempt was made to leach the raw massive sulphide ore to recover the minerals of chief commercial value in the Flin Flon ore. The o nly commercially available oxidizers of sulphides minerals that will make them soluble in sulphuric acid, are atmospheric oxygen and ferric sulphate. It was found that the zinc

blende and the chalcopyrite dissolved hardly at all in 36 hours in sulphuric-acid-ferric sulphate solution, at room temperature. Hot solutions would increase the yield, but were considered commercially impracticable.

Roasting and Leaching.

To roast an ore like the Flin Flon, containing chalcopyrite, zinc blende and pyrite, the temperature used during roasting "must be a compromise that will give a reasonable recovery of the first two, (which in themselves should be roasted at different temperatures) without making too much of the iron soluble (16)".

Two methods of leaching were tried:- (1) leaching by percolation, and (2) leaching by countercurrent decantation.

Leaching by percolation requires smaller volumes of solutions to wash out the base-metal solutions entrained in the calcine, yields a rich pregnant liquor, and a residue which would be in a suitable mechanical form for cheap treatment by cyanidation for recovery of precious metals.

In a countercurrent system, water could be used for a neutral-leach which would take into solution the water-soluble zinc. Acid formed by the hydrolysis of ferric and cupric salts, would dissolve zinc oxides. The neutral-leach curcuit could be made the one for the extraction of the zinc and the acid-leach,

made acid by addition of sulphuric acid, the one for the extraction of the copper.

Roasting tests were made in the laboratory 30-inch MacDougall-type furnace, electrically heated. The depth of bed on each hearth was from 1 to 1 1/4 inches.

The results of Roasting Test No. 1. (Table No. 4.) show that the amounts of the copper and the zinc soluble in acid would be satisfactory commercially, but not the amounts soluble in water. If a satisfactory amount of the zinc and the copper could be made water soluble, this would eliminate the iron, and very likely prevent some impurities from entering the solution and interfering in the extraction of the copper and the zinc.

The temperatures employed in Roasting Test No. 1. was evidently slightly too high to give best results for the copper and too low for the zinc. Test No. 2. was attempted to see if the chalcopyrite could be sulphated and removed from the more difficultly oxidizable zinc blende by roasting at a low temperature. The results obtained are given in Table No. 4.

Addition of common salt was found to lower the temperature for formation of sulphates. Three per cent of salt was added to fifth hearth. The results of the test show that addition of salt lowered the sulphide sulphur in the calcine

from about o.5 per cent to 0.2 per cent, but the water-soluble iron was not changed appreciably (Test No. 3. Table No. 4.)

In leaching tests, the calcine was pulped with 5 per cent solution of H_2SO_4 for the acid-leach, and with water for the neutral-leach. The extractions of leaching tests are given in Table No. 4.

Treatment of Residue from Leached Products.

The leached products from the original ore left behind a residue that had to be treated further for precious metals. The precious metals were recovered by cyanidation. From 33 to 59 per cent of the gold and from 56 to 82 per cent of the silver were recovered with heads carrying 0.13 ounces of gold and 1.3 ounces of silver per ton of ore. Better recoveries were obtained with acidulated brine,- (2 or 5 parts of brine solution to 1 of residue)carrying 1 per cent of bleaching powder and o.9 per cent of sulphuric acid. This solution gave an extraction of 89.5 per cent of gold and 73.8 per cent of silver. On agitating for 8 hours, up to 91.7 per cent of gold and 80.6 per cent of silver were recovered.

Conclusions on Roasting and Leaching.

As a result of the tests on roasting the following conclusions were drawn by the Denver Laboratories,- "In roasting the original Flin Flon ore under conditions employed,

it is not believed to be feasible to alter the copper minerals only so the blende may be removed. Neither is it possible to sulphate both the zinc and the copper to such an extent that water-leaching alone may be used and the dissolving of many other impurities thus avoided. In so roasting the unconcentrated Flin Flon ore for subsequent leaching with sulphuric acid that both the zinc and the copper may be made soluble to a satisfactory extent and yet the amount of acid-soluble iron may be under one per cent, the operating staff, at least, believe that such a treatment is not practicable with the best than can be done by present roasting methods (17)."

It was found that in leaching by percolation, air pockets were formed and led to uneven percolation and caused a slow rate of percolation of the leach liquors.

To keep a neutral-leach for the zinc and a separate acid-leach for the copper, it would be necessary to have a high degree of sulphation of the zinc (and the copper). Any zinc which was not removed in the neutral-leach, would be picked up in the acid-leach and thus would give considerable difficulty in recovery of copper.

For deposition of the zinc from the neutral-leach and the deposition of the copper from the acid-leach, it would be necessary to remove the impurities.

Tests on cyanidation of the leached residue showed that recovery of gold and silver was possible. But before the cyanide was added, the copper and the acid would have to be removed thoroughly in order to prevent excessive cyanide consumption.

In conclusion, it may be said that, because the ratio of soluble impurities to valuable constituents in the roasted calcine is high, many difficulties are brought about and passed on to the other stages of the process,- leaching, solution purification and electrolysis,- that the scheme as a whole would cost too much to compete with the practice of concentration plus smelting.

Flotation Concentration.

It was, therefore, determined to resort to flotation to make copper concentrates, and zinc concentrates; these concentrates containing as much of the precious metals as possible. It was planned to treat the tailings for their precious metal content, by cyanidation, if they were rich enough to justify this treatment. Most of the tests were made on the massive sulphide ore. The results of the tests are recorded in Table No. 1.

SUMMARY OF FLOTATION TESTS.

Flotation tests were principally on the flotation reagents to determine what reagents and the amounts would yield the highest recovery and grade of concentrates of the valuable

minerals.

Reagents for the Removal of Talc.

The disseminated ore contains an appreciable amount of talcy gangue which has to be removed by flotation prior to flotation of the sulphide minerals. Experimental tests have shown that the talcy material is readily floated by frothing agents. Aldol ($CH_3.CH(OH).CH_2.CHO$), which is a good frothing, non-collecting agent, yielded a 5.7 per cent copper loss and 0.8 per cent copper in the talc froth.

Various fumols were tried. Fumol No. 21., with 4 pounds of soda ash yielded a 4.9 per cent copper loss in the talc froth. The same amount of fumol No. 21. with 6 pounds lime increased the copper in the talc froth from .90 per cent to 3.80 per cent, and the copper loss from 4.9 per cent to 46.1 per cent.

Various fumols have been tried; most of them yielding high copper values and losses in the talc froth. Only fumols Nos. 20 and 21 in pulps containing soda ash and fumol No. 25 in pulp containing lime yielded a low loss of copper of about 5 per cent in the talc froth, but fumol No. 20 yielded a relatively high copper value.

The General Engineering Company attempted to remove the talc by frothing it with pine oil. .05 pounds of pine

oil yielded a talc froth, running o.45 per cent copper, a loss of 3.2 per cent. Pine oil gave the lowest copper loss in the talc froth and was adopted as the talc removal in the flotation practice for the Flin Flon ore.

Reagents for the Recovery of the Copper Minerals.

Lime was added to the ball-mill to give alkalinity to the pulp. Varying amounts of the lime were added, ranging from 2 pounds to 40 pounds per tone of ore. In some tests, the amount of the lime added was very small. This was due to the fact that the water from the tailings, which was still alkaline, was used again in the flotation circuit; as in the case of the tests by the Denver Laboratories.

Decreasing the lime from 40 pounds to 24 pounds, with 5 pounds of x-y reagent (4 parts x-cake, 6 parts xylidin), concentrates increased from 14.2 to 16.7 per cent copper, and the recovery of copper increased from 78.5 per cent to 87.9 per cent. The zinc content in the copper concentrate did not change, but the copper loss in the zinc concentrate increased from 7.7 per cent to 10.6 per cent. By decreasing the x-y reagent to o.3 pounds and lime to 16 pounds, a higher grade of copper concentrate was obtained, running 17.4 per cent copper and 87.4 per cent recovery, and the loss of the zinc in the copper concentrates was decreased from 10.6 per cent to 7.4 per cent. By decreasing

the x-y reagent 0.3 pounds the silver value in the copper concentrates increased to 2.64 ounces per ton of ore. The silver value increased in the copper concentrate with the decrease of lime from 1.38 to 2.14 ounces silver per ton of ore. If it is desired to recover the precious metals from the ore, with the copper concentrates, the amount of the lime added to the ball-mill will have to be low.

The General Engineering Company used x-cake (naphthylamine a) as a collecting agent, and Wattson's residue, as a frothing agent, for the copper minerals. With 0.1 pounds of x-cake, 4 pounds of Wattson's residue (equivalent to 0.6 pounds of oil) and 14 pounds of lime, a recovery of 83.1 per cent copper was obtained, but the grade of the concentrates was much lower than with x-y reagent, being only 8.00 per cent copper. By increasing the x-cake to 0.15 pounds and Wattson's residue to 6 pounds, the recovery didn't change, but the grade of the copper concentrates decreased to 6.70 per cent while the zinc increased from 5.6 to 7.8 per cent.

Tenth of a pound of thio-fizzan with 2 pounds lime yielded an average of 17.0 per cent copper and an average recovery of 82.6 per cent of copper concentrates from the disseminated ore. Increasing the thio-fizzan to 0.25 pounds and the lime to 8 pounds, increased the grade of the copper concentrate to 18.68 per cent copper, but the recovery decreased to 76.0 per cent.

0.15 pounds TT mixture (thiocarbanilid dissolved in ortho-toluidin, 1:4), which is a collecting agent, with 6 pounds lime gave a recovery of 80.9 per cent and 18.8 per cent copper in the copper concentrate. The same amount of TT mixture with 4 pounds soda ash yielded, practically, the same copper concentrate, but a lower recovery (74.5 per cent). As the cost of the soda ash is higher than that of the lime, and the lime, besides giving alkalinity to the pulp, acts as a depressant for the pyrite, the lime was adopted as the conditioning reagent. Increase of the TT mixture to 4 pounds, the lime to 8 pounds and using Aldol as a frother, the grade of copper concentrate decreased to 14.7 per cent.

The Denver Laboratories, for the concentration of the sulphide ore, with 4 pounds of lime, 1 pound of soda ash, 2 pounds sodium sulphite, 0.10 pound sodium xanthate and 2 drops of pine oil, obtained a concentrate of 17.0 per cent copper, a recovery of 78.6 per cent, and a zinc loss of 6.0 per cent in the copper concentrate. By decreasing the lime to 2.7 pounds and increasing the sodium xanthate to 0.20 pounds and omitting the soda ash and sodium sulphite, but using 1.2 pounds sodium chloride (assumed to act on the talc, giving a cleaner copper concentrate), a copper recovery of 81.5 per cent copper concentrate was obtained, running 17.0 per cent copper, 5.0 per cent zinc, and a zinc loss in the copper

concentrates of 9.0 per cent.

The above reagents yielded satisfactory results and were adopted as standard for subsequent flotations.

<u>Reagents for the Recovery of the Zinc Minerals</u>.

Recovery of the zinc minerals, in the Flin Flon sulphide ore, was first tried by the Ore Dressing and Metallurgical Laboratories, Ottawa. 40 pounds of lime added to the ball-mill, 10 pounds soda ash, 1 pound gravity fuel oil No. 34, yielded a low recovery of the zinc. It was thought that by dewatering the copper tailings, the slight depressing effect of the lime on the zinc blende may be somewhat eliminated. In the subsequent tests the copper tailings were dewatered. The same amount of lime and gravity fuel oil No. 34, 8 pounds soda ash, increased the grade of the zinc concentrate from 29.0 per cent to 36.5 per cent, and the zinc recovery from 44.1 per cent to 58.3 per cent, when the copper tailings were dewatered. The copper loss in the zinc concentrate increased from 3.3 per cent to 8.4 per cent.

By decreasing the lime to 24 pounds, and addition of 8 pounds of soda ash, 3 pounds of Wattson's residue 1-B, 1 pound copper sulphate (activator for the zinc blende), the grade of the zinc concentrate increased to 40.7 per cent, but the recovery decreased to 52.5 per cent, and the loss of 21.8 per cent zinc in

the tailings. By further decrease of lime to 20 pounds in the ball-mill, and addition of 5 pounds of lime, 8 pounds soda ash, 1 pound fuel oil, an increase of 3.8 per cent zinc was obtained, but the recovery decreased by 10 per cent approximately. Ten pounds of caustic soda substituted for 5 pounds of lime decreased both the grade of the zinc concentrates and the recovery; increasing the zinc loss in the tailings to 24.8 per cent.

The General Engineering Company obtained a 60.3 per cent zinc recovery, using 14 pounds of lime in the ball-mill, 6 pounds of sulphuric acid, 6.5 pounds Wattson's residue, 1 pound G.E. Co. No. 209 and 0.1 pound G.E. Co. No. 56. By decreasing the Wattson's residue to 0.75 pound and leaving out the sulphuric acid, the zinc recovery decreased to 54.8 per cent.

The Denver Laboratories, using 2 pounds of soda ash, 1 pound sodium sulphite, 1 pound of copper sulphate, 0.10 pound xanthate and 1 drop of cresylic acid as frother, obtained a fairly high zinc concentrate (43.5 per cent), a recovery of 78.5 per cent. By substituting 0.5 pound of lime for soda ash and increasing the copper sulphate to 1.6 pounds and xanthate to 0.24 pounds, the grade of the zinc concentrate increased to 46.0 per cent and the zinc loss in the tailing decreased from 15.5 to 10.5 per cent.

CONCLUSIONS ON CONCENTRATION BY FLOTATION.

Since leaching, and roasting and leaching methods

were not feasible for the concentration of the Flin Flon sulphide ore, and the concentration by flotation gave satisfactory recovery of both the copper and the zinc, it was considered that flotation concentration was the only commercially feasible method which could be employed.

Before any definite conclusions could be drawn as to possible commercial success of the concentration, larger scale operations were required.

EXPERIMENTAL TWO-TON PLANT (18).

To confirm the results of the laboratory experiments on flotation a small pilot plant with a capacity of two tons per day was erected at the laboratories in Denver, based on flow sheet shown in Figure 6.

On July 1st, 1926, several carloads of the Flin Flon ore were received, and the plant was put in operation. Difficulties were immediately encountered due to the high specific gravity of the ore, making it necessary to add so much water to keep the pulp flowing in the launders and flotation machine passages that the flotation circuit became too dilute to give good results. The circuit was rebuilt to conform to the flow sheet shown in Figure 7, and after these changes were made, good results were obtained. (See Table No. 1. Test No. 1. by Denver Laboratories).

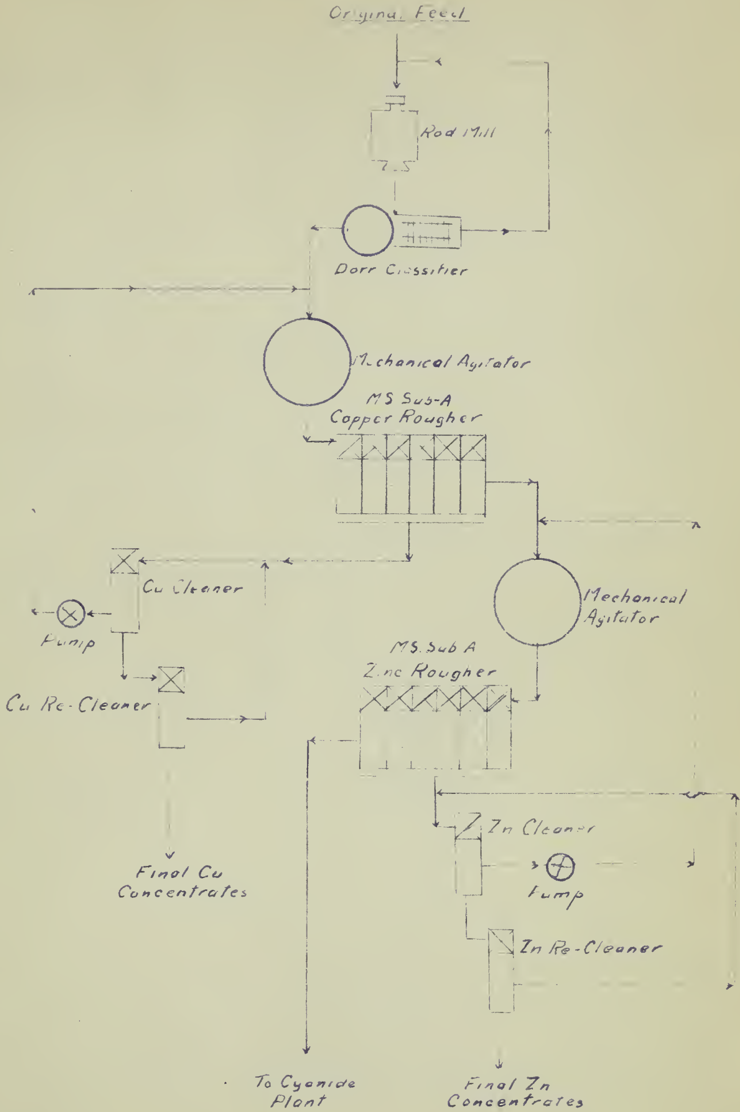


FIG. 6. FLOW SHEET OF TWO-TON PILOT MILL
DENVER, COLORADO.

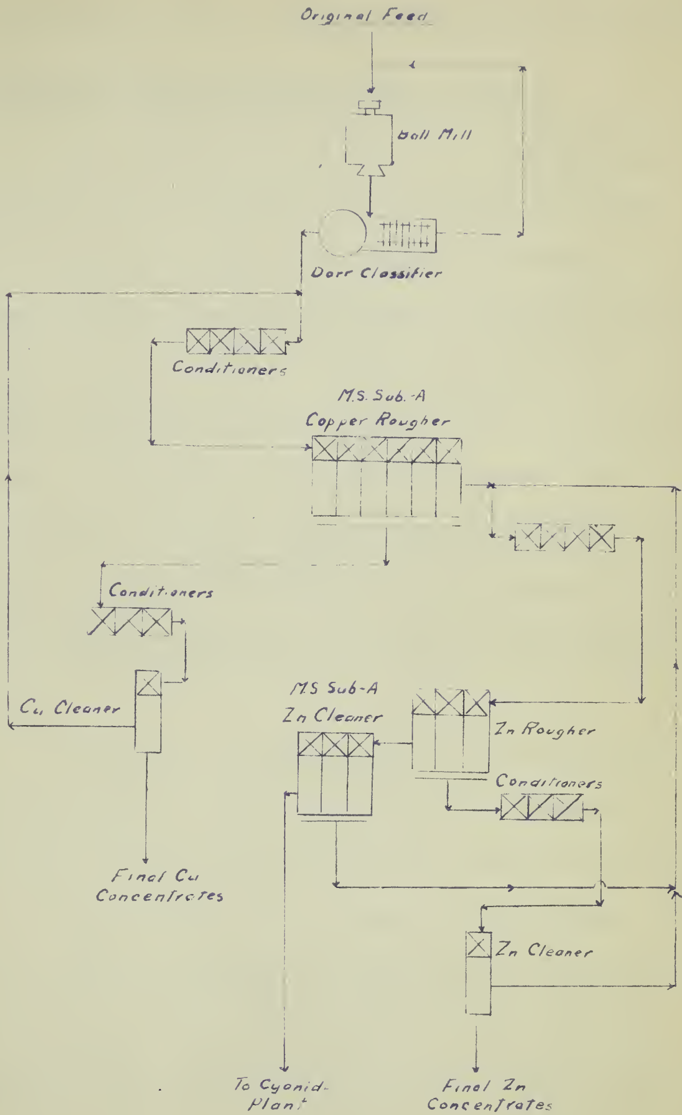


FIG. 7 FLOW SHEET OF MODIFIED
TWO-TON PILOT MILL
DENVER, COLORADO.

The tailings from the two-ton mill were treated in a small cyanidation plant for the recovery of gold and silver.

25-TON FLOTATION PLANT AND 10-TON CYANIDE PLANT.

The results obtained from the 2-Ton plant showed that Flin Flon ore could be concentrated by flotation processes for the recovery of copper and zinc; and that precious metals, not picked up in these circuits could be recovered by subsequent cyanidation of the tailings from the zinc circuit. On August 1st, 1926, it was decided to install a 25-Ton flotation plant at the mine, based on the flow sheet Figure 8, with a 10-Ton cyanide plant for treating the flotation tailings. The mill was continuously in operation from March 22nd, to 31st, 1927, The average of the results obtained are shown in Table No. 1, (Flotation tests 25-Ton Pilot Mill).

Although the results showed a low extraction of the copper and low grade of the copper concentrates they were considered fair in view of the small amount of the copper in the feed.

These results were obtained on the massive sulphide ores, but as mining proceeded, disseminated ore was encountered. The talcy mineral in the disseminated ore gave a low grade copper concentrate and a low recovery of the copper, due to the talcy mineral floating off in the copper concentrate cells along with the copper. The zinc section was not greatly affected.

Mine development, at that stage, indicated that a close separation could not be made between the disseminated talcy ore and the massive sulphide ore in the mine. The flotation work showed that even a small amount of the talc (as the talcy mineral was called) would seriously interfere with the treatment of the sulphide ore.

The talcy mineral has the property of absorbing a certain part of the oils and reagents in considerable amounts, so that the promoter effect of the xanthate is largely overcome, unless a large amount is used, and hence there is little selective action between the chalcopyrite and the pyrite.

Tests were again started both in the laboratory and in the mill in order to devise a process to, either overcome the effect of the talc or remove it before the regular flotation treatment.

Test results showed that it was impossible to overcome the absorbing effect of the talc by using starch in a neutral or acid circuit. Also, it was practically impossible to maintain exactly right alkalinity and add the exact amount to the mill when the character of the feed varied. The use of starch was discontinued and a talc flotation substitued. A large percentage of the talc was removed in a low-grade product by adding pine oil to the alkaline pulp and floating the talc. The flow sheet of the pilot mill was changed to accommodate the talc removal as shown

in flow sheet, Figure 9.

By removing the clean concentrates from the first two rougher cells in each circuit a better extraction was obtained,- "as the coarse mineral that floated in these cells was reluctant to float again when sent to a cleaner, and it was found that the concentrates from these roughers were of better grade than the concentrates formerly produced when all the froth was cleaned (19)". By returning the middling froth and cleaner tailings to the third rougher cell, a higher grade zinc concentrate was obtained than by returning to the original feed of the zinc circuit. Regrinding the zinc middlings lowered the tailings about 0.2 per cent zinc.

On September 1st, 1927, a mixture of disseminated and sulphide ores was fed to the mill. The results given in Table No. 1. were obtained for a considerable time. Disseminated ore alone was treated giving satisfactory recovery of the copper and grade of concentrates. (See Table No. 1.)

The mill was operated until March 10th, 1928, treating ore from various parts of the mine, and trying various changes in the flow sheets, reagents, etc., but it was found that the reagents and flow sheet of September 1st, 1927, gave the best results on both classes of ore. The Mineral Separation talc cells were substituted for the Fagergren machine, due to lower maintenance cost and the advantage of having all cells of one type.

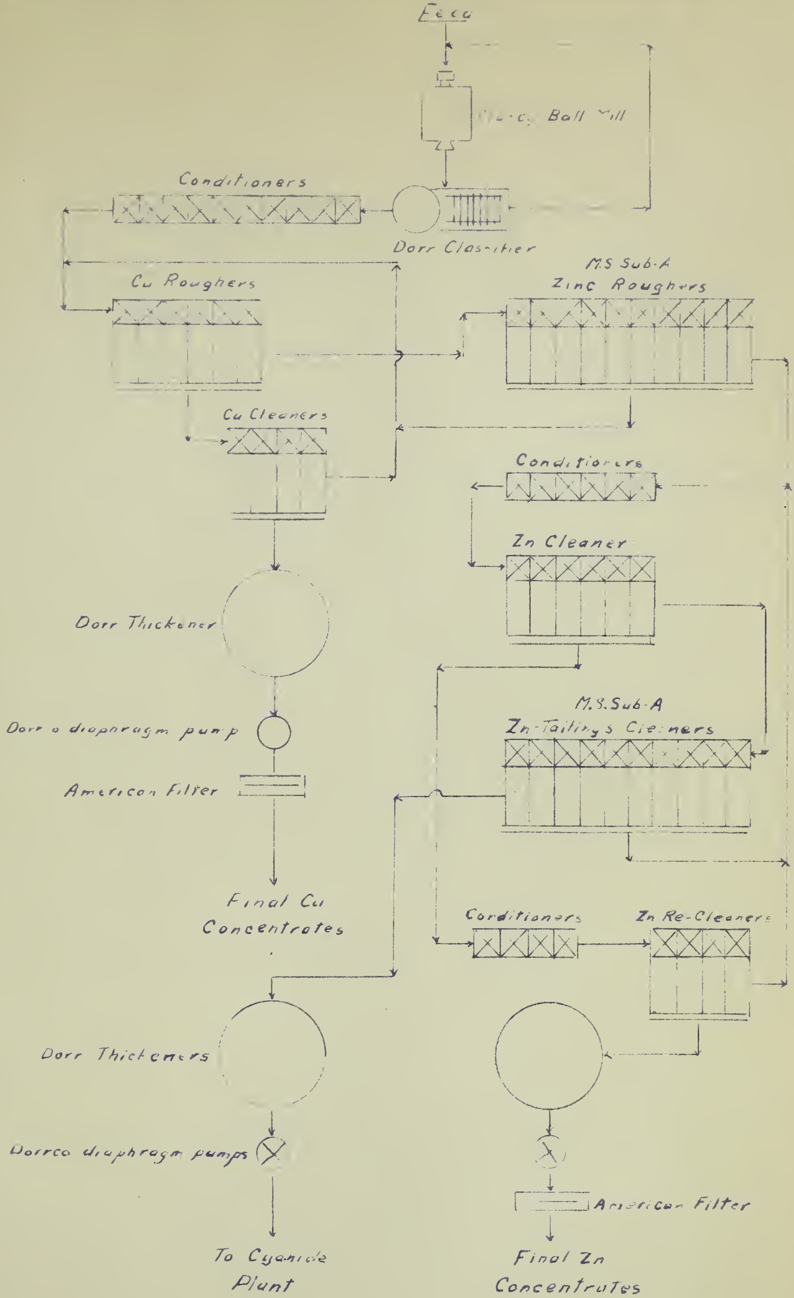


FIG.8. FLOW SHEET OF 25-TON FLOTATION PLANT
FLIN FLON MAN.

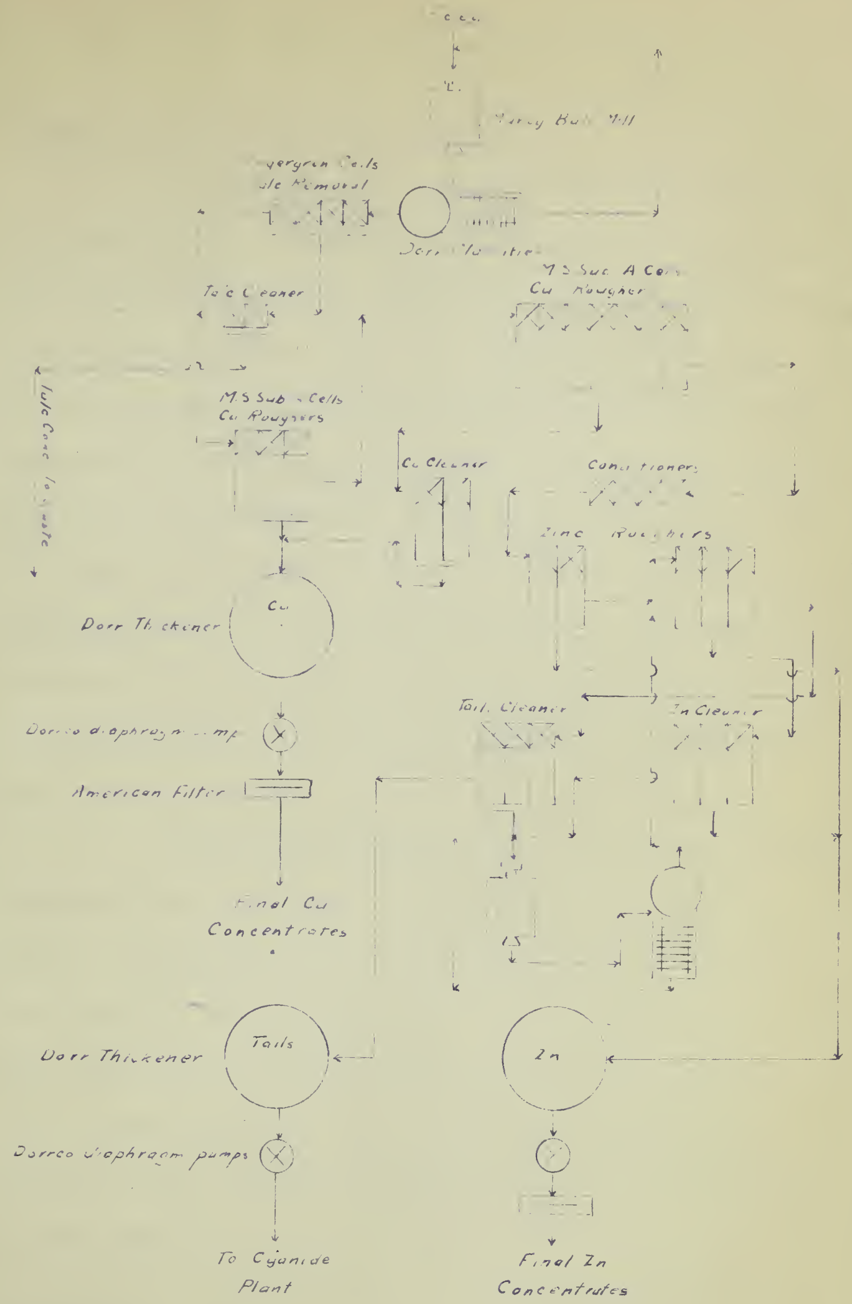


FIG 9. FLOW SHEET OF MODIFIED 25-TON PILOT PLANT
FLIN FLON MAN

SUMMARY OF RESULTS OF 25-TON PILOT MILL.

The 25-Ton mill, operating continuously for a considerable period, treating a mixture of sulphide ore and disseminated ore, yielded a copper concentrate of 14.28 per cent copper, a recovery of 72.5 per cent. The copper loss was 15.7 per cent in the talc froth and tailings, and 11.8 per cent in the zinc concentrate. The zinc concentrate obtained ran 43.2 per cent zinc, a recovery of 81.6 per cent. The zinc loss was 6.8 per cent in the copper concentrate, and 11.6 per cent in the talc froth and tailings. Almost half the gold and over one-third of the silver was removed with the copper concentrates; but 40.7 per cent of the gold and 45.1 per cent of the silver remained in the talc froth and the tailings.

A continuous separation and concentration of the disseminated ore yielded a copper concentrate of 18.0 per cent copper, a recovery of 91.2 per cent; but 33.9 per cent of the zinc floated with the copper froth. No recovery of the zinc was made. The gold in the talc froth and tailings was 29.3 per cent, being appreciably less than for the mixture of sulphide and disseminated ores. The silver in the talc froth and tailings was 43.7 per cent.

CONCLUSIONS.

As the results of the work at the 25-Ton pilot mill,

the following requirements would have to be considered in order that the flotation operations would yield satisfactory results;-

1. Fine grinding to liberate the mineral particles; avoiding too excessive sliming. (85 per cent - 200 mesh - 65 per cent - 300 mesh).
2. Removal of sufficient talc; the amount depending on the feed.
3. Careful control of alkalinity throughout the circuit. (PH of 9.0 to 9.4 in the copper circuit; 11.6 in the zinc rougher circuit).
4. Proper pulp density in the flotation circuit, adding as little fresh water as possible to the circuit after the pulp leaves the classifier. (1.3 specific gravity or 30 per cent solids).
5. Proper time of conditioning after adding reagents. (From 3.5 to 4 minutes at least ahead of the zinc float).

Various other reagents were tried, but the reagents of September 1st, 1927, gave the best results on both types of ore.

An attempt was made to refloat the zinc tailings in an acid circuit and obtain a concentration of gold which would eliminate the cyanide plant, but this was unsuccessful.

THE 3000-TON MILL.

Following the 25-Ton pilot mill, it had been definitely proven, that the concentration for the copper and the zinc;and the cyanidation for the precious metals in both types of Flin Flon ore, was commercially feasible. Designing and construction of a 3000-ton capacity mill was immediately started. A detailed description of the 3000-ton mill was given in C.I.M. & M. Bulletin, No. 221, September, 1930, Concentration and Cyanidation at Flin Flon Pilot Mill.

The flow sheet, Figure 10, will give some idea of the mill.

This mill was put into operation gradually in June, 24th, 1930, coming into full production in November 19th, 1930, and has operated continuously since that time.

SUMMARY AND CONCLUSIONS OF PREVIOUS EXPERIMENTAL WORK.

The experimental tests, conducted by the Ore Dressing and Metallurgical Laboratories, Ottawa, the General Engineering Company, and the Denver Laboratories, show that the only commercially feasible method of concentrating the Flin Flon ore is by froth flotation. Roasting and leaching cannot be commercially employed because the ratio of soluble impurities to valuable constituents in the roasted calcine is high. The impurities bring up and

Stephens Adams Tripper

7 Steel Fine Ore Bins
150 tons capacity each

7 - ['66 Hardinge Ball Mill
35 tons ... 35 ...

7 ... S... P...

1 ...
6 ... Dorr Classifiers

Distributors 2

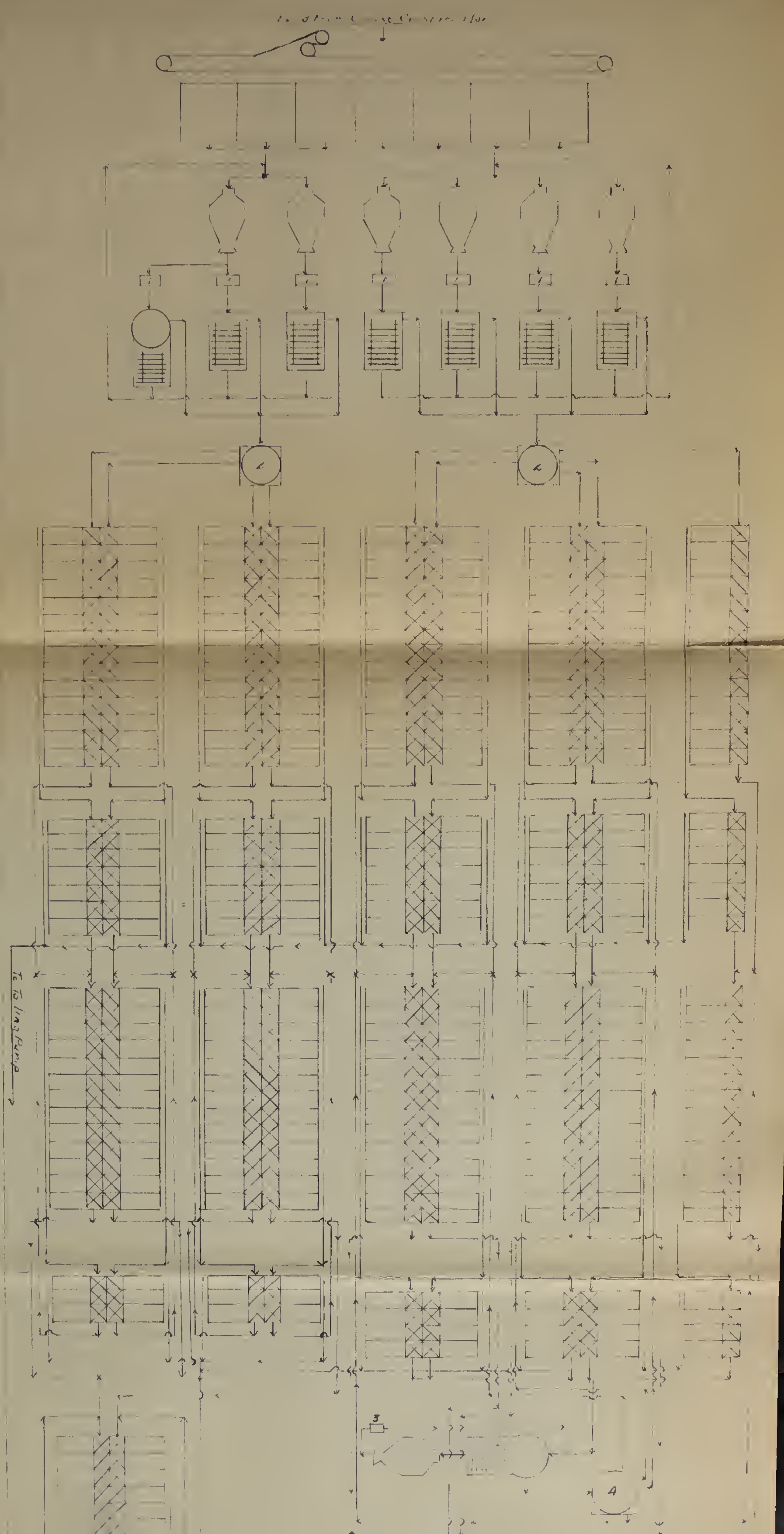
... 24 M.S. Sub A Machines
(... units for disseminated ore section
5 units for solid sulphide ore section)

7-24 ... Sub A Machines
Talc Cleaners

Copper Roughers
13 24" M.S. Sub A Machines
for disseminated ore section
14 24" M.S. Sub A Machines
for solid sulphide section

Copper Cleaners
... 24 M.S. Sub-A Machines
... 24 M.S. Sub-A Machines

1 Dorr bowl Classifier
10'x66" Hardinge Ball Mill
3½" Wilfley Sand Pump #3
Distributor #4



1 Dorr bowl Classifier
10 x 66" Hardinge Ball Mill
3½" Wilfley Sand Pump # 3
Distributor # 4

Zinc Roughers
6 - Conditioner Cells (1 unit)
7 Conditioner Cells (6 units)
12 24" MS Sub-A Mach. (1 unit)
14 24" MS Sub-A Mach (6 units

Zinc Cleaners
1 Conditioner Cell (7 units
4-24" M.S. Sub-A Mach. (6 units)
5 24" MS Sub-A Mach. (1 unit)

Dorr Thickener (traction-type)

2 4 Duplex Dorrco diaphragm pump #5
3 Wilfley Sand Pumps # 6
Dorr bowl Classifier
10 x 66" Hardinge Ball Mill
Wilfley Table
Devereaux Agitators #7
3" Wilfley Sand Pump # 8
4" duplex Dorrco diaphragm pump #9
Dorrco Filters # 10

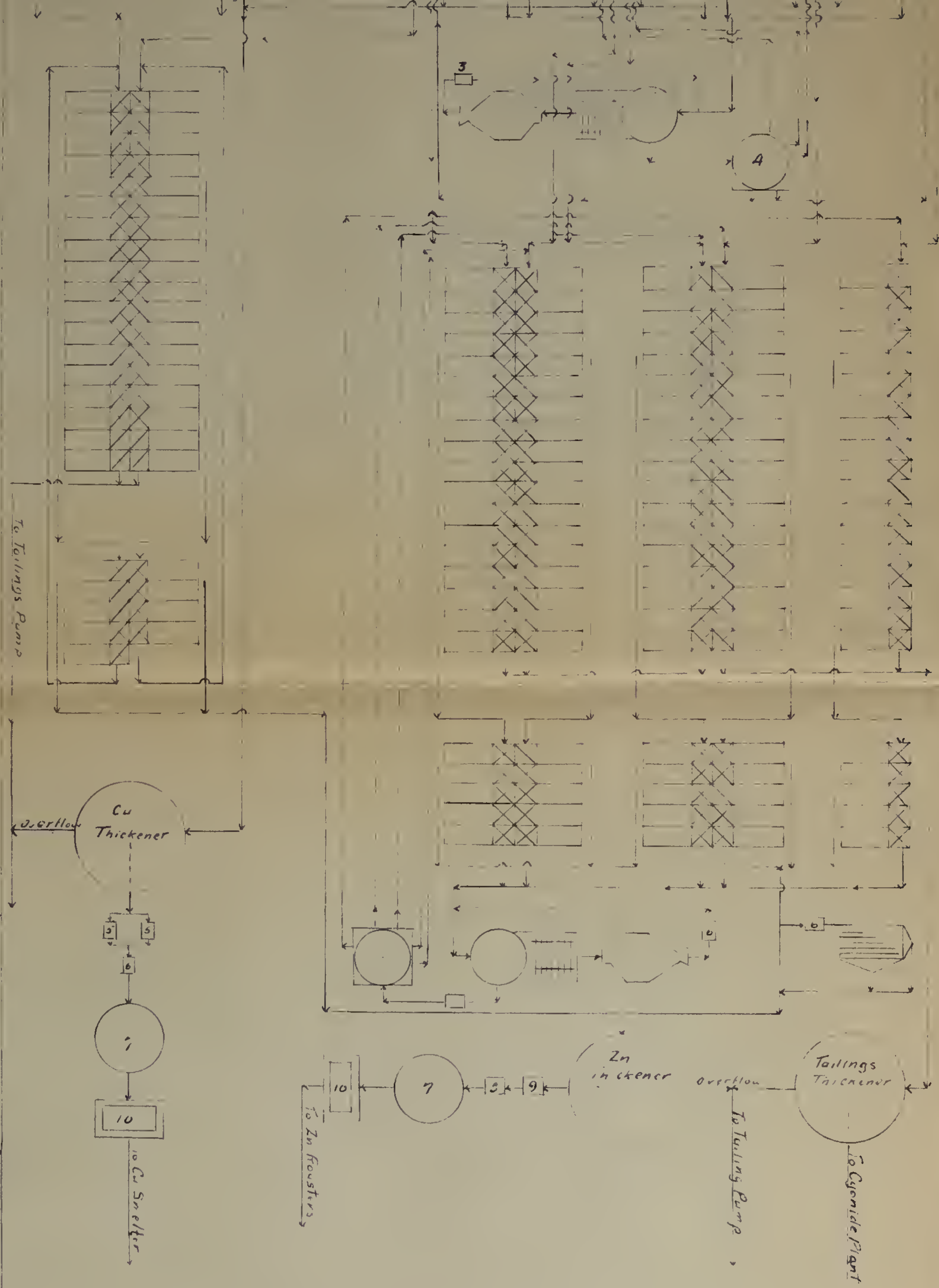


FIG 10. FLIN FLON CONCENTRAOR FLOW SHEET
FLIN FLON, MAN.

pass on many difficulties to the other stages of the process,- leaching, solution purification and electrolysis,- that the scheme as a whole would cost too much to compete with the practice of concentration by flotation followed by subsequent smelting.

Pyritic smelting cannot be employed as the ore consists of two types,- the massive sulphide and the disseminated. The latter is too low in sulphide minerals for direct smelting practice.

Jigging and table concentration of disseminated ore gave low recovery of the valuable minerals. Considerable copper went as slimes along with the tailings.

Concentration by flotation gave satisfactory recovery of the copper and the zinc for both types of ore; and was thought to be the only commercially feasible method which could be employed.

Experimental tests on the concentration of the Flin Flon disseminated ore by flotation indicated that the talcy mineral would have to be removed before the copper and the zinc minerals were recovered. The talcy gangue is readily floated by frothing agents. The frothing agent must be non-collecting in order that the sulphide minerals would not be floated with the talc froth. Pine oil, used for the removal of talc, yielded a low copper loss

in the talc froth. In the 25-ton pilot mill, using pine oil, a froth was obtained, running 0.4 per cent copper, 0.76 per cent zinc, a copper loss of 3.7 per cent and a zinc loss of 9.98 per cent. Several fumols have been tried, but the results were not as satisfactory as those obtained with pine oil or Aldol. Pine oil gave the lowest copper loss in the talc froth and was adopted as the talc removal at the Flin Flon concentrater.

Varying amounts of lime were added to the ball-mill. It was found that the amount of lime required was that amount which would give a hydrogen ion concentrate of 9.0 to 9.4 in the copper circuit. Higher alkalinity of the pulp in the zinc circuit was necessary. A pH of about 11.4 was maintained in the zinc circuit by addition of lime.

Lime has a slight depressing effect on the chalcopyrite minerals. A decrease of lime from 40 pounds to 24 pounds increased the grade of copper concentrate from 14.2 per cent to 16.7 per cent copper and the recovery from 78.5 per cent to 87.9 per cent. A decrease in the lime increased the silver values.

A better recovery of the precious metals in the copper concentrates was obtained with x-cake than with x-y reagent. As high as 4.5 ounces of silver and 0.26 ounces of gold per ton of copper concentrates was obtained; but the x-cake yielded a lower grade of copper concentrates.

Thio-fizzan was used for copper recovery in the concentration of the disseminated ore. Increasing the thio-fizzan from 0.10 pound to 0.25 pound and the lime from 2 to 8 pounds increased the copper concentrates from 17.0 to 18.68 per cent copper, but the recovery decreased from 82.6 to 76.0 per cent. 0.15 pound of TT mixture yielded practically the same grade of copper concentrates as the thio-fizzan, but the recovery was higher.

Sodium sulphite, used as a depressant for zinc, yielded a zinc loss of 6.0% in the copper concentrates. Zinc sulphate substituted for sodium sulphite yielded practically the same zinc loss.

A continuous operation of several days treating a mixture of heavy sulphide ore and disseminated ore, with 2.02 pounds lime, 2.00 pounds of zinc sulphate added to the ball-mill, 0.063 pound pine oil added to talc cells, 0.09 pound xanthate to copper cells, yielded 14.28 per cent copper, and 4.1 per cent zinc in the copper concentrate, a recovery of 72.5 per cent copper. 49.6 per cent of the gold and 35.5 per cent of the silver was recovered with the copper concentrates. The same reagents used in treating a disseminated ore, yielded 18.0 per cent copper concentrate, a recovery of 91.2 per cent; 70.66 per cent of the gold and 56.27 per cent of the silver was recovered

with this concentrate.

Dewatering the copper tailings for zinc flotation increased the zinc in the concentrates from 29.0 to 36.5 per cent and the recovery from 44.1 to 58.3 per cent.

Various reagents have been tried in the zinc circuit. It has been found that copper sulphate (activator for zinc minerals) yielded the best results. Addition of 0.74 pound copper sulphate, 2.00 pounds lime, 0.15 pound cresylic acid, 0.12 pound xanthate to zinc rougher cells, and 0.18 pound copper sulphate, 0.55 pound lime to zinc cleaner cells yielded a concentrate of 43.2 per cent zinc, a recovery of 81.6 per cent, and a loss of 11.6 per cent zinc in the tailings.

The following reagents,- 2.02 pounds lime, 2.00 pounds zinc sulphate added to the ball-mill; 0.063 pound pine oil to talc cells; 0.09 pound aerofloat, 2.81 pounds sodium chlorite, 0.04 pound xanthate to copper cells; 0.74 pound copper sulphate, 2.00 pounds lime, 0.15 pound cresylic acid, 0.12 pound xanthate to zinc rougher cells; 0.18 pound copper sulphate, 0.55 pound lime to zinc cleaner cells yielded a satisfactory recovery of the base metals and the precious metals during a continuous operation in a 25-Ton Pilot Mill and were adopted as standard flotation reagents at the Flin Flon mill.

TABLE NO.1. FLOTATION TESTS.

Test No.	Flotation Reagents in Lbs. Per Ton of Ore.	Product	Wt %	Analysis Cu %	Analysis Zn %	Analysis Au oz.	Analysis Ag oz.	Recovery or Loss Cu %	Recovery or Loss Zn %	Recovery or Loss Au %	Recovery or Loss Ag %	Remarks.
	ORE DRESSING AND METALLURGY, DEPARTMENT OF MINES, OTTAWA, 1921.											
14	40 lbs.lime to ball-mill; 5 lbs.x-y reagent to Cu cell 10 lbs.soda ash; 1 lb.No.34 gravity fuel oil to Zn cell.	Cu Conc	14.1	14.2	4.4	0.10	1.38	78.5	10.2			Ruth Machine.
		"Midd.	15.3	2.3	5.9	0.14	1.27	13.8	14.9			
		Zn Conc	9.3	0.9	29.0	0.16	0.90	3.3	44.1			
		" Midd	6.3	0.5	6.3	0.10	1.10	1.2	6.5			
		Tails	54.0	0.15	2.7			3.2	24.3			
15	40 lbs.lime to ball-mill; 5 lbs.x-y reagent to Cu cell; 8 lbs.soda ash; 1 lb. No.34 gravity fuel oil to Zn cell.	Cu Conc	16.0	12.5	2.7		1.54	77.9	7.7			Dewatered Copper Tailing.
		" Midd	11.0	1.4	7.0		0.90	6.0	13.6			
		Zn Conc	9.0	2.4	36.5		1.70	8.4	58.3			
		" Midd	10.0	0.8	5.5		0.40	3.1	9.8			
		Tails	54.0	0.22	1.1		0.71	4.6	10.6			
16	24 lbs.lime to ball-mill; 4 lbs.x-y reagent to Cu cell; 1 lb.P.T.T.Co.No.1580	Cu Conc	10.4	17.8	3.6	0.10	2.14	69.2	6.0	15.8	21.4	Dewatered Copper Tailing.Unsuccessful Zinc Recovery.
		" Midd	14.5	3.6	7.2	Tr.	2.00	19.6	16.8	4.2	28.0	
		Tails	75.1	0.4	6.4	0.07	0.70	11.2	77.2	80.0	50.6	
19	24 lbs.lime to ball-mill; 5 lbs.x-y reagent to Cu cell; 8 lbs.soda ash,3 lbs. Wattsons residue 1-B,1 lb. copper sulphate,1 lb. No.34 gravity oil.	Cu Conc	14.2	16.7	4.4	0.10		87.9	10.6	18.2		Dewatered Copper Tailing.
		" Midd	2.8	1.8	8.7	0.10		1.9	4.2	3.6		
		Zn Conc	7.6	0.45	40.7	0.11		1.3	52.5	10.8		
		" Midd	7.2	0.5	8.9	0.07		1.3	10.9	6.6		
		Tails	68.2	0.3	1.9	0.07		7.6	21.8	60.8		
21	16 lbs.lime to ball-mill; 0.3 lb.x-y reagent to Cu cell.	Cu Conc	13.8	17.4	3.2	0.08	2.64	87.4	7.37	13.8		Dewatered Copper Tailing. Zinc flotation unsuccessful.
22	20 lbs. lime to ball-mill; 0.3 lbs.x-y reagent to Cu cell; 5 lbs. lime; 8 lbs. soda ash;1 lb.fuel oil and pine oil to Zn cell.	Cu Conc	12.9	15.7	2.7	0.12	2.58	73.7				Dewatered Copper Tailing.
		" Midd	4.6	2.6	6.8	0.12	1.84					
		Zn Conc	6.1		44.5				45.2			
		" Midd	11.6	0.58	6.7	0.08	1.18					
		Tails			2.8							
32	20 lbs.lime to ball-mill; 0.3 lbs.x-y reagent to Cu cell; 10 lbs.caustic soda,8 lbs.lime,1 lb.fuel oil to Zn cell.	Cu Conc	19.5	8.80	2.4			67.8	7.9			Dewatered Copper Tailing.
		" Midd.	9.4	2.35	4.3			8.7	6.8			
		Zn Conc	9.8	3.42	26.3			13.2	43.5			
		" Midd	13.7	0.70	7.3			3.8	16.9			
		Tails	47.6	0.35	3.1			6.5	24.8			
34	40 lbs.lime to ball-mill; 0.2 lb.x-y reagent to Cu cell;6 lbs.soda ash; 1 lb. gravity fuel oil to Zn cell	Cu Conc	18.0	10.9	3.5	0.10		65.5	11.6			Dewatered Copper Tailing.
		" Midd	8.3	7.75	4.9	0.08		21.2	7.2			
		Zn Conc	8.6	2.30	35.0	0.15		6.6	54.1			
		Zn Midd	8.6	1.05	7.8	0.06		3.0	11.9			
		Tails	56.5	0.20	1.5			3.7	15.2			
	GENERAL ENGINEERING COMPANY, SALT LAKE CITY, UTAH, 1921.											
4	14 lbs.lime,0.1 lb.x-cake, 4 lbs.Wattsons 1-B to Cu cell; 6 lbs.H_2SO_4,6.5 lbs. Wattsons 1-B,1 lb.G.E.Co.No. 209,0.1 lb.G.E.Co.No.56 to Zn Cell	Cu Conc		8.00	5.6	0.26	4.50	83.1				Callow Cell.
		Zn Conc		1.25	37.6	0.08	2.12		60.3			
		Tails		0.22	1.3	0.07	0.90					
14	14 lbs.lime,0.15 lb.x-cake, 6 lbs.residue 1-B to Cu Cell 1 lb.G.E.Co.No.209,0.75 lb. Wattsons 1-B,0.1016.G.E.Co. No.56 to Zn cell.	Cu Conc		6.70	7.8	0.22	3.90	83.4				Callow Cell.
		Zn Conc		1.20	38.5	0.08	1.75		54.8			
		Tails		0.25	0.24	0.07	0.80					
	ORE DRESSING AND METELLURGY, DEPARTMENT OF MINES, OTTAWA, 1922.											
2	0.10 lb.thio-fizzan,4 lb. lime.	Conc	12.7	16.9				85.6				Ruth Machine.Ground wet. 65 mesh.
		Midd.	16.5	1.75				7.4				
		Tails	70.8	0.25				7.0				
3	Same as Test No.2.	Conc.	19.9	11.63				85.7				These figures are the results of flotation test only.Talc removed. 100 mesh.
		Midd.	15.9	1.00				5.9				
		Tails	64.2	0.35				8.4				
11	Same as Test No.2.	Conc.	12.5	13.78				67.3				*hydraulic classifier slimes. 100 mesh.
		Midd.	9.8	2.00				7.7				
		Tails	59.5	0.53				12.4				
		Slimes	13.5	1.05				5.6				
12	Same as Test No. 2.	Conc.	16.3	11.16				76.8				Recovery of Table conc. & flot.conc. 82.4% Test No.12.was duplicate of Test No.11., with the exception that separated slimes were tabled. *Table conc.,*Table tails. 100 mesh.
		Midd.	14.7	1.35				8.4				
		Tails	51.7	0.30				6.5				
		Conc.*	2.5	5.30				5.6				
		Tail.*	8.4	0.75				2.7				
13	Same as Test No. 2.	Conc.2	21.4	9.92				81.7				No. 1. were products of slime flotation. Grade of conc.from two conc. 8.71% Cu. 100 mesh.
		" 1	4.5	2.74				4.7				
		Midd.2	17.4	1.12				7.4				
		Tail.2	49.2	0.27				5.1				
		" 1	3.2	0.88				1.1				
15	2 lbs. lime; Aldol to talc cell. o.10 lb.thio-fizzan to Cu cell.	Conc.	11.4	18.92				81.1				200 mesh.
		Midd.	11.4	1.42				6.0				
		Tails	58.0	0.33				7.2				
		Froth	19.2	0.80				5.7				
21b	Same as Test No. 15.	Conc.	14.0	15.87				83.3				
		Midd.	11.3	1.10				4.7				
		Tails	54.7	0.22				4.5				
		Froth	20.0	0.96				7.3				
22b	Same as Test No. 15.	Conc.	12.6	16.27				81.0				
		Midd.	11.8	1.25				5.8				
		Tails	61.7	0.34				8.3				
		Froth	13.9	0.90				4.9				
26	5 lbs.lime, 0.10 thio-fizzan	Conc.	15.5	11.60				64.6				-48 to -200 mesh. 44 per cent -200 mesh.
		Midd.	17.6	2.27				18.1				
		Tails	66.9	0.72				17.3				
27	Same as Test No. 26.	Conc.	14.5	13.62				76.8				-65 to -200 mesh 53 per cent -200 mesh.
		Midd.	14.6	1.50				7.5				
		Tails	70.9	0.57				15.7				
28	Same as Test No. 26.	Conc.	11.9	18.67				86.4				-100 tp -200 mesh 70 per cent -200 mesh.
		Midd.	13.4	0.40				2.0				
		Tails	74.7	0.40				11.6				
29	Same as Test No. 26.	Conc.	14.0	14.92				80.1				-100 to -200 mesh 82 per cent -200 mesh.
		Midd.	15.0	1.57				9.0				
		Tails	71.0	0.40				10.9				
31	8 lbs. lime; TT mixture	Conc.	10.8	20.22				83.0				-100 to -200 mesh 82 per cent -200 mesh.
		Midd.	9.0	2.15				7.3				
		Tails	80.2	0.32				9.7				
33	Lime and YZ mixture	Conc.	11.2	18.02				77.3				-100 to -200 mesh.
		Midd.	2.06	2.00				15.7				
		Tails	68.3	0.25				6.6				
36		Conc.1	11.6	20.42								1000 gram was floated. The middling from the recleaning of the conc from the first lot was
		Tails1	88.4	0.45								
		Conc.2	12.5	16.40								
		[illegible]	[illegible]	[illegible]								

		Midd.	15.0	1.57				9.0				82 per cent -200 mesh.
		Tails	71.0	0.40				10.9				
31	8 lbs. lime; TT mixture	Conc.	10.8	20.22				83.0				-100 to -200 mesh 82 per cent -200 mesh.
		Midd.	9.0	2.15				7.3				
		Tails	80.2	0.32				9.7				
33	Lime and YZ mixture	Conc.	11.2	18.02				77.3				-100 to -200 mesh.
		Midd.	2.08	2.00				15.7				
		Tails	68.3	0.25				6.6				
36		Conc.1	11.6	20.42								1000 gram was floated. The middling from the recleaning of the conc from the first lot was added to the second lot and flotation repeated as for first lot. This was repeated five times.
		Tails1	88.4	0.45								
		Conc.2	12.5	16.40								
		Tail.2	87.5	0.85								
		Conc.3	11.4	19.05								
		Tail.3	88.6	0.60								
		Conc.4	9.4	16.15								
		Tail.4	90.6	1.15								
		Conc.5	8.2	12.15								
		Tail.5	91.8	1.75								
38	6 lbs. lime; TT mixture.	Conc.	15.4	13.08				75.3				Callow cell. * Clean up cells.
		Midd.*	3.9	4.55				6.5				
		Tails	80.7	0.60				18.2				
37b	Same as in Test No. 15.	Conc.	14.7	15.78				83.3				Callow cell. * Clean up cells.
		Midd.*	6.2	1.25				2.8				
		Tails.	57.2	0.35				7.2				
		Froth	21.9	0.85				6.7				
44	Fumol No. 21. to talc cell; 4 lbs. soda ash, 0.15 c.c. TT mixture.	Froth	14.4	0.90				4.9				Ruth machine.
		Conc.	10.6	18.55				74.5				
		Midd.	12.6	2.85				13.5				
		Tails	62.4	0.30				7.1				
45	6 lbs. lime; 3 drops fumol No. 6. to talc cell; 0.15 c.c TT mixture.	Froth	24.3	1.12				10.3				Ruth machine.
		Conc.	10.0	20.27				78.4				
		Midd.	13.7	1.75				9.3				
		Tails	52.0	0.10				2.0				
46	As 44 but Fumol No. 20.	Froth	9.6	1.45				5.3				Ruth machine.
		Conc.	12.1	16.95				78.6				
		Midd.	1.68	1.68				9.2				
		Tails	64.1	0.28				6.9				
47	As 45, but Fumol No. 21.	Froth	31.7	3.80				46.1				Very viscous and floats too much gangue.
		Conc.	6.8	16.75				43.4				
		Midd.	8.4	1.95				6.2				
		Tails	53.1	0.21				4.3				
48	Fumol No. 22a.	Froth	30.9	1.60				18.7				
		Conc.	7.3	24.43				68.1				
		Midd.	11.1	2.05				8.8				
		Tails	50.7	0.23				4.4				
49	Fumol No. 23.	Froth	21.8	1.40				11.7				
		Conc.	8.5	20.93				70.0				
		Midd.	15.1	2.18				13.0				
		Tails	55.2	0.23				5.3				
50	Fumol No. 24.	Froth	23.8	0.80				7.2				
		Conc.	11.3	16.80				80.9				
		Midd.	14.0	1.70				9.1				
		Tails	50.0	0.15				2.8				
51	Fumol No. 24a.	Froth	22.4	1.45				12.5				
		Conc.	9.8	21.40				77.1				
		Midd.	13.0	1.48				7.7				
		Tails	54.8	0.13				2.7				
52	Fumol #25.	Froth	13.8	1.05				5.4				
		Conc.	11.1	19.50				80.2				
		Midd.	14.5	1.57				8.4				
		Tails	60.6	0.27				6.0				
53	Fumol No. 26.	Froth	20.3	2.12				16.0				
		Conc.	9.8	21.05				72.6				
		Midd.	15.1	1.32				7.4				
		Tails	49.8	0.20				4.0				
54	Fumol No. 27.	Froth	9.9	6.09				21.8				
		Conc.	14.2	12.40				64.4				
		Midd.	15.6	1.52				8.7				
		Tails	60.2	0.25				5.1				

GENERAL ENGINEERING COMPANY, SALT LAKE CITY, UTAH, 1922.

1	.05 lb. pine oil to talc cell; 8 lbs. lime; 0.4 lb. TT mixture; .0833 lb. Aldol.	Conc.	14.5	14.74				80.1				Callow cell; 80% -200 mesh.
		Tails	66.5	0.67				16.7				
		Talc	19.0	0.45				3.2				
2	.05 lb. pine oil to talc cell; 8 lbs. lime; 0.25 lb. thio-fizzan.	Conc.	10.9	16.68				76.2				Callow cell; 80% -200 mesh
		Tails	65.5	0.31				19.8				
		Talc	23.6	0.44				4.0				

DENVER LABORATORIES, DENVER, COLORADO.

1	4 lbs. lime to ball-mill; 1 lb. soda ash; 2 lbs. sodium sulphide; 0.10 lb. Na xanthate 2 drops pine oil to Cu cell; 2 lbs. soda ash; 1 lb. Na sulphide; 1 lb. Cu sulphate; 0.10 lb. Na xanthate, 1 drop cresylic acid to Zn cell.	Heads		1.45	3.36	0.08	0.87					M.S. Sub-A machine 85% -300 mesh. Floated for 15 minutes.
		Cu Conc		17.0	3.0	0.53	4.93	78.6	6.0	50.0	38.3	
		Zn "		1.1	43.5	0.03	1.59	4.6	78.5	2.2	11.2	
		Tails		0;28	0.6	0.05	0.50	16.8	15.5	53.8	50.5	
2)	2.7 lbs. lime to ball-mill; 1.2 lbs. NaCl, 0.20 lb. Na xanthate, 1 drop pine oil to Cu cell. 0.5 lb. lime, 1.6 lbs. Cu sulphate, 0.24 lb. xanthate, 1 drop cresylic acid to Zn cell.	Heads		1.89	4.61	.099	1.36					Floated 5 minutes. 1 lb. lime to Zn cleaner cell.
		Cu Conc		17.0	5.0	.55	5.3	81.5	9.0	50.0	35.3	
		Zn "		1.4	46.0	.03	1.5	5.8	79.5	2.0	9.4	
		Tails		0.3	0.6	.06	.94	12.7	10.5	48.0	55.3	

TWO-TON PILOT MILL, DENVER LABORATORIES, DENVER, COLORADO.

1	Same as Test No. 2. above	Cu Conc		18.0	5.0							
		Zn "		1.2	44.0							
		Tails		0.25	0.6							

25-TON PILOT MILL, FLIN FLON, MANITOBA, 1927.

	No record given.	Cu Conc	3.88	6.31	5.3	0.42	14.02	42.7	2.5	59.6	28.9	Continuous run from March 22 to March 31.
		Zn "	16.9	1.01	41.3	0.29	3.20	29.7	83.7	17.9	29.7	
		Tails	79.2	0.20	1.45	0.08	1.01	27.6	13.8	22.5	42.4	
		Feed		0.57	8.3	.027	1.88					
	Ball-mill:- 2.02 lbs. CaO; 2.00 lbs. Zn sulphate. Talc cell-0.063 lb. pine oil. Cu cell; 0.09 lb. aerofloat, 2.81 lbs. NaCl, 0.04 lb. xanthate. Zn cell; 0.74 lb. Cu sulphate 2.00 lb. CaO, 0.15 lb. cresylic acid 0.12 lb. xanthate.	Cu Conc	8.7	14.28	4.1	0.32	3.70	72.5	6.8	49.6	35.5	Sept. 1st. 1927 Mixture of sulphide ore and disseminated ore. Continuous operation. Zn cleaners:-0.18 lb. Cu sulphate, 0.55 lb. CaO.
		Zn "	9.8	2.05	43.2	.055	1.78	11.8	81.6	9.7	19.4	
		Tail & Talc Fr	81.5	0.33	0.7	.028	0.50	15.7	11.6	40.7	45.1	
	Same as above.	Cu Conc	10.5	18.0	3.95	0.12	1.57	91.2	33.9	70.7	56.3	Disseminated ore. Continuous operation.
		Talc Fr	16.0	0.4	0.76	.005	0.20	3.1	10.0	4.5	11.0	
		Tails	73.5	0.16	0.93	.006	0.13	3.7	56.1	24.8	32.8	
		Heads		2.06	1.22	.018	0.29					

TABLE NO.2. GRAVITY FLOTATION.

Test No.	Products	Weight %	%-Cu	Recovery %	Remarks
3	Conc.	22.1	4.15	36.02	Concentrates, middling and tailing were mixed together and retreated by flotation (Test No.3. Table No.1.) The <u>froth</u> was the collection of the talc gangue mineral which floated in the settling tank. The <u>loss</u> is the discrepancy between the amount fed and that recovered in the products. 13.21% of Cu was lost in froth, slimes and loss.
	Midd.	29.1	2.60	29.70	
	Tailing.	31.6	1.70	21.07	
	Froth*	3.0	2.50	2.90	
	Slimes.	7.9	2.48	7.73	
	Loss*	6.3	-	2.58	
7	Conc.	26.8	5.10	51.1	
	Midd.	14.5	4.10	22.2	
	Tailing	40.0	0.94	14.1	
	Slimes	18.7	1.80	12.6	
8	Conc.	39.3	5.00	73.8	The ore was not crushed as fine as in Test No.7. in the hope that less copper would be lost in the slimes. The loss was 3.8 per cent less.
	Tailing	46.1	1.00	17.3	
	Slimes	13.1	1.78	8.8	
11	Flot.Conc.	12.5	13.78	67.2	The reagents used in flotation were thio-fizzan and lime.
	Flot.Midd.	9.8	2.00	7.7	
	Flot.Tail.	59.5	0.53	12.4	
	Slimes	13.5	1.05	5.6	
	Loss	4.7	-	6.9	
12	Flot.Conc.	16.3	11.16	76.8	Recovery in table conc. and flot.conc. 82.4% Test No.12 was a duplication of Test.No. 11. with the exception that the separated slimes were tabled making conc. and tailing. Reagents used in flot. were thio-fizzan and lime.
	Flot.Midd.	14.7	1.35	8.4	
	Flot.Tail.	51.7	0.30	6.5	
	Tble Conc.	2.5	5.30	5.6	
	Tble Tail.	8.4	0.75	2.7	
	Loss	6.4	-		
13	Flot.Conc.2	21.4	9.92	81.7	Products No.1. were the slime flotation. Products No.2 were the sand flotation. Grade of conc. from two conc. 8.71% Cu.
	Flot.Conc.1	4.5	2.74	4.7	
	Flot.Mid.2	17.4	1.12	7.4	
	Flot.Tail2	49.2	0.27	5.1	
	Flot.Tail1	3.2	0.88	1.1	
	Loss.	4.3			

TABLE NO.3. TEST NO.36.

Product	Weight grams	Copper	
		Per Cent	Per Cent x grams
Concentrate No. 1.	108	20.42	22.06
Tailing No. 1.	820	0.45	3.69
			25.75
Concentrate No. 2.	117	16.40	19.18
Tailing No. 2.	822	0.85	6.99
			26.17
Concentrate No. 3.	107	19.05	20.39
Tailing No. 3.	833	0.60	5.00
			25.39
Concentrate No. 4.	105	18.15	19.06
Tailing No. 4.	1,011	1.15	11.63
			30.69
Concentrate No. 5.	81	12.15	9.84
Tailing No. 5.	910	1.75	15.93
			25.77

TABLE NO. 4. ROASTING AND LEACHING TESTS.

Test No.	Time Hrs.	Temp. Deg. C.	Products	Analysis					H_2O Solu. %			Acid Solu. %			Remarks
				Cu-%	Zn-%	Fe-%	Au-oz	Ag-oz	Cu	Zn	Fe	Cu	Zn	Fe	
1	7.5	850	Calcine	1.54	5.1				69	80	1.7	93	92		Roasting Test.
			Ore	1.37	4.3	36.5	0.10	1.28							
2	5	500-530	Calcine	1.54	5.0		0.17	1.33	71	36	2.5	84	42	2.5	Roasting Test.
			Ore	1.33	4.5		0.10	1.08							
3		620-675	Calcine						85	88	1.7	93	95		3 per cent NaCl added
4			Calcine									87	92	1.7	Leaching by Percolation
5			Calcine									79	98	0.9	" " "
6			Calcine						62	83					Leaching by Countercurrent decantation.
7			Calcine									71.8	72	12.2	

CHAPTER 4.

PRESENT INVESTIGATIONS.

The results of the operation of the 25-Ton Pilot mill determined the flotation requirements as follows (Table No.5).

TABLE NO.5.

Reagents.	Lbs.Per Ton of Ore.
To Ball Mills.	
Lime	2.02
Zinc sulphate	2.00
To Talc Cells.	
Pine oil	0.063
To Copper Cells.	
Aerofloat	0.09
Salt (NaCl)	2.81
Xanthate	0.04
To Zinc Roughers.	
Copper sulphate	0.74
Lime	2.00
Cresylic acid	0.15
Xanthate	0.12
To Zinc Cleaners.	
Copper sulphate	0.18
Lime	0.55
Total,	10.763

The total amount of reagents required for the separation and concentration of the Flin Flon ore, according to the above table, is considerably larger than that required by average mill practice on copper-zinc flotations where the copper mineral is chalcopyrite and the zinc mineral is zinc blende.

Four plants operating on chalcopyrite-zinc blende ore report the following amounts of reagents per ton of ore treated (Table No. 6.). -

TABLE NO. 6.

Reagents added to	25-Ton Pilot Mill, Flin Flon	Rouyn(20)	Amulet(21)	Sherritt-Gordon(20)	Arizona (20)
	Amounts in pounds per ton of ore				
Ball Mill	4.02	3.50	0.92	5.15	5.80
Talc Cells	0.063	-	-	-	-
Cu Cells	2.94	0.48	0.25	0.56	0.05
Zn Cells	3.74	3.08	3.39	1.58	2.20
Total,	10.763	7.06	4.56	7.29	8.05

It is apparent that the reagent consumption is excessive and should be open to considerable improvement.

Grade of concentrates and recoveries obtained at the plants, reported in Table No. 6., are given in Table No. 7.

This table shows clearly that the concentration procedure at Flin Flon, even with the excessive reagent consumption, is not all that could be desired.

The copper concentrate produced carries only a little better than 14 per cent copper while good average practice should yield at least 20 per cent . 14 per cent concentrate, undoubtedly, is expensive to smelt, as the large amounts of impurities will

greatly increase the fuel and flux requirements, besides, increasing handling costs and reducing furnace capacities. Moreover a copper recovery of only 72.5 per cent presents a considerable waste in an ore which is lean in copper at any time.

The zinc concentrate produced, carries over 43 per cent zinc; this may be considered a reasonably high grade concentrate. Recovery of zinc in the concentrate, however, amounts to only 81.6 per cent, the balance going to the copper concentrate, the talc froth and the tailings, with the majority (11.6 per cent) in the last two mentioned products. This is more than twice that reported at the other plants listed in Table No. 7.

Gold and silver recoveries in the copper concentrate are low. Only 49.6 per cent of the gold and 35.5 per cent of the silver are recovered as compared to 80.0 per cent of the gold and 89.0 per cent of the silver at Rouyn, and 75.0 per cent of the gold and the silver at Amulet. Over 40 per cent of the gold and silver present inthe ore at Flin Flon is lost to the tailings. To recover some of this, the tailings from the heavy sulphide section is treated by cyanidation, but precious metal values in the talc froth and in the tailings of the disseminated ore section are an accepted loss in the mill practice.

[illegible]

[illegible]

TABLE NO. 7.

Products	25-Ton Pilot Mill Flin Flon, Man.				Rouyn Quebec				Amulet Quebec				Sherritt-Gordon Man.				Arizona	
	ANALYSIS PER CENT (Au & Ag in Oz. per Ton)																	
	Cu	Zn	Au	Ag	Cu	Zn	Au	Ag	Cu	Zn	Au	Ag	Cu	Zn	Au	Ag	Cu	Zn
Cu Conc.	14.28	4.10	.320	3.70	23.08	4.58	0.08	7.60	24.0				17.70	3.62	0.16	3.50	8.0	7.5
Zn "	2.05	43.20	.055	1.78	1.01	25.17	0.03	0.75		52.0			0.63	36.30	0.10	0.27	1.2	54.0
Talc & Tails	0.33	0.74	.028	0.50	0.23	0.34	Tr.	0.26					0.08	0.33	0.01	0.07	0.16	1.0
Heads	1.71	5.20	.056	0.90	7.36	6.53	.03	2.45	3.5	13.0	.02	3.50	2.00	5.51	0.04	0.48	0.60	7.0
	PER CENT RECOVERIES OR LOSSES.																	
Cu Conc.	72.5	6.8	49.6	35.5	95.6	20.9	80.0	89.0	92		75	75	92.5	6.9	64.0	80.0	56.0	4.5
Zn "	11.8	81.6	9.7	19.4	2.8	76.5	19.9	5.9		80			4.0	88.6	4.0	9.8	21.0	83.5
Talc & Tails	15.7	11.6	40.7	45.1	1.6	2.6		5.1					3.5	4.5	32.0	11.1	23.0	12.0

It is thus apparent that the flotation practice at the Flin Flon mill, if based on the 25-Ton Pilot plant operations, is open to considerable improvement.

Possibilities of Improvement.

The flotation operation developed by the laboratory and pilot plant operations is a straight selective or differential flotation process with a four stage separation,- talc - copper - zinc and tailings.

There is one very definite alternative scheme possible, namely, a so-called bulk flotation, in which the sulphides are floated from the gangue minerals in a single operation and then the bulk concentrates treated by selective or differential flotation for the separate minerals. By such a scheme, the mixed ore (sulphides and disseminated ores) would be subjected to a rough concentration whereby the copper, zinc and iron minerals would be separated from the gangue minerals and thus could be arranged to yield a tailings (the gangue minerals) with minimum metal values. This would be the final waste product of the mill; all other intermediary products being sent back into concentration. The relatively low grade sulphide concentrate would be treated separately by selective flotation to yield copper, zinc, and iron concentrates.

In this scheme of operation there are two possible ways to handle the talc fro m the disseminated ore;- (1) allow it to collect with the bulk float concentrate and removed subsequently, or (2) make a preliminary talc separation prior to bulk float of the sulphides.

There are several apparent advantages from bulk flotation;

1. The first operations are solely to make a clean tailings,- i.e., one with minimum of metal values. Total recoveries would be immediately controllable by this operation and should tend to much higher recoveries of the metal values of the ore.
2. Removal of from 25 to 50 per cent of the ore as clean tailings should result in considerable reduction in reagent consumption, as the less bulk of material to be treated will require less total reagents to make the selective flotation concentrates.
3. All metal values are kept in circulation in the cleaning circuits until such time as they report to one or other of the concentrates. Hence the grade of the concentrates produced should be controllable within close limits.

With these apparent improvements as a possible result

of bulk flotation operations, it was decided that some investigation on the suitability of the scheme was warranted. Three hundred pounds of the Flin Flon ore were obtained ~~and~~ from the mill. As received, this consisted of fine ore bin material, and was almost entirely disseminated ore.

Procedure.

Any investigation on flotation involve the study of the effects of variations in a very large number of factors, any one of which may materially alter the efficiencies of the operations. Exact comparison of results of experiments are only possible when all conditions are, as nearly as possible, the same for each set of experiments. It is, therefore, impossible to compare or correlate different experimental results unless the ore used comes from the same shipment, the water comes from the same source, and the reagents used are of the same origin. Under these conditions comparison of bulk flotation experiments could only be made with selective flotation experiments, done on the same materials and under, as approximately as possible, the same conditions.

To check the results of bulk flotation, therefore, it was necessary to establish the efficiency of the selective flotation operation in the same laboratory.

The present investigations, therefore, had to be divided

into two parts.

1. Extensive selective flotation experiments, using essentially the reagents developed in the pilot mill operations, to determine the University of Alberta laboratory equivalent of the pilot mill operations; or at least to establish maximum recovery conditions for the laboratory when operating on the particular sample of ore obtained.
2. To develop a bulk flotation procedure which would be comparable with the selective flotation results obtained inthe laboratory.

Direct comparison of the results of these two series of experiments on a recovery-cost basis should indicate the possible savings, if any, obtainable by the use of bulk flotation operations.

Equipment.

Ore as received at the laboratory was crushed in a 4" x 6" Dodge type laboratory Jaw Breaker (Figure 11.) to 3/8", and re-crushed in an Allis Chalmers sample grinder (Figure 12.) to 28 mesh Tyler. It was then stored in air-tight cans until required for experimental purposes. Grinding prior to flotation was done in a Denver Laboratory Weinig type ball-mill (Figure 13.) capable of both, wet and dry, grinding. All

flotation runs were made in a Laboratory type Mineral Separation Sub-A machine (Figure 14.).

FIG. 11 DODGE JAW-BREAKER

FIG. 12 ALLIS-CHALMERS SAMPLE GRINDER

FIG.13 DENVER LAB. WEINIG TYPE BALL MILL

FIG.14 M.S. SUB-A MACHINE

CHAPTER 5.

METHODS OF CHEMICAL ANALYSES

The products obtained, from the experimental work on the concentration of the Flin Flon ore by froth flotation, were talc froth, copper concentrates, zinc concentrates, iron concentrates and tailings. The materials determined in each product were, copper, zinc, iron and insolubles. Also gold and silver values were determined in some flotation products. Analyses for each product were made in duplicates.

Analysis for Copper by Potassium Iodide Method.

This method depends upon the fact that cupric salts when treated with potassium iodide liberate iodine. The reaction is as follows:-

$$2\ CuSO_4 + 4\ KI = 2CuI + 2\ K_2SO_4 + I_2$$

The liberated iodine is titrated with standard thiosulphate solution.

$$2\ Na_2S_2O_3 + 2\ I = Na_2S_4O_6 + 2\ NaI$$

When the free iodine has entirely vanished, the blue color (produced by starch with free iodine) will change to white, which is the end-point.

The method used in the determination of copper in

the flotation products was the "Short Iodide Method (22)". This method is based on the repression of ionization of iron by the addition of sodium fluoride to an acetic acid solution containing the ferric salt. The iron fluoride, Fe_2F_6, formed does not react with KI. Ag, As, Bi, Cd, Co, Fe, Hg, Mn, Mo, Ni, Pb, Sn, U, Zn, do not interfere. Cr forms an insoluble sulphate, which holds Cu; hence H_2SO_4 should be avoided when Cr is present. Vanadium interferes and when present, this method cannot be used.

This method was adopted for the determination of the copper in the flotation test products because it is a very rapid method and gives most accurate results and is most practical. A.H. Low, when speaking of electrolytic method, and iodide method for copper determination, states,- "I, therefore, give the iodide method the preference in most cases as being more practical and nearly if not quite as accurate as the electrolytic at its best (23)".

Standard Solutions and Standardization. Sodium thiosulphate solution was made by dissolving 19.5 $\pm$ grams of salt in 1 litre of distilled water. The solution was kept in the dark when not being used, to prevent changing in strength.

To standardize the thiosulphate solution, 0.2 grams of

copper foil was weighed, placed in a beaker and dissolved in 10 c.c. of 1:1 HNO_3. The mixture was then diluted to 25 c.c., boiled till red fumes ceased to come off, removed and diluted to 75 c.c., NH_4OH added till just alkaline, and acetic acid added till solution was acid. The solution was then cooled and 15 c.c. of KI solution added (1 gm.KI in 5 c.c. of water). The mixture was titrated till yellow color of free iodine was nearly gone and then 5 c.c. of starch solution added; then titrated again till blue color of starch iodide disappeared. The thiosulphate solution was standardized before using.

Starch solution was made by stirring 5 grams of starch in to 200 c.c. of cold distilled water. It was then poured into 800 c.c. of boiling water and boiled until clear. 10 grams of NaCl was added to prevent the starch from changing and will thus keep indefinitely. The solution was allowed to settle and the clear liquid poured off for use.

<u>Procedure for Copper Determination.</u> A 0.5 gram sample was weighed into a 250 c.c. beaker, decomposed with 15 c.c. HCl, 5 c.c. HNO_3 and a little $KClO_3$. When the reaction ceased 5 c.c. of H_2SO_4 was added and fumed on a hot plate practically to dryness. The solution was then cooled, 10 c.c. of water and 2.c.c.

of bromine water added, and boiled to expell excess bromine.

While the solution was still hot, 20 - 25 c.c. of ammonium acetate was added to precipitate all iron as $Fe(C_2H_3O_2)_3$ and 2 to 5 grams of solid Na_2F_2 (till red color changed to bluish green or white). The mixture was then cooled and when cold 15 c.c. of KI solution was added and titrated with standard sodium thiosulphate solution, using starch as indicator.

Analysis for Iron and Insolubles by "Permanganate Method (22)".

The method depends upon the quantitative oxidation of ferrous salts to the ferric condition when potassium permanganate is added to their cold solution; the following reaction taking place:

$$10\ FeSO_4 + 2\ KMnO_4 + 8\ H_2SO_4 = 5\ Fe_2(SO_4)_3 + K_2SO_4 + 2\ MnSO_4 + 8\ H_2O.$$

Potassium permanganate produces an intense pink color in solution, so that it acts as its own indicator.

Reasons for choice of this method were that it gave rapid determinations, sharp end point, and accurate results.

Standard Solutions and Standardization. Potassium permanganate solution was made by dissolving 5.26 grams of crystals in one litre of distilled water. The solution was kept in a colored bottle to prevent decomposition.

To standardize the permanganate solution, o.2 grams of sodium oxalate was dissolved in 250 c.c. of water. To the solution 10 c.c. of H_2SO_4 was added and heated to 70° C. The solution was titrate hot to permanent pink tinge.

Procedure for Iron and Insolubles Determinations. A 0.5 gram sample was weighed into 250 c.c. beaker, 2 c.c. of water and 10 c.c. HCl added. When reaction ceased 5 c.c. HNO_3 was added and boiled till brown fumes ceased. The solution was then cooled and when cold 10 c.c. of 1.1 H_2SO_4 was added and fumed strongly to white fumes. The sample was then cooled, 10 c.c. of water added, boiled to dissolve sulphates, filtered (while hot) and washed 5 or 6 timeswith hot distilled water.

The residue was ignited for insolubles.

To the filtrate a strip of aluminum was added and boiled to reduce the ferric sulphate to ferrous sulphate. When the solution was coloress it was cooled and the aluminum strips washed, the solution filtered ~~was~~ then diluted to 250 c.c. and treated with standard potassium permanganate solution. The first permanent pink tinge was taken as the end-point.

Analysis for Zinc by Ferrocyanide Method.

The method used for the determination of zinc in the flotation products was slightly modified from the "regular method"

as outlined by A. H. Low (24).

This method depends upon the zinc precipitation by potassium ferrocyanide from acid solutions as insoluble zinc potassium ferrocyanide.

$$3\ ZnCl_2 + 2\ K_4Fe(CN)_6 = K_2Zn_3(Fe(CN)_6)_2 + 6\ KCl.$$

Although the procedure is not very short and rapid, it was adopted because of the fair degree of accuracy of results, if proper procedure is followed. Low values may result when iron, prepipitated as ferric hydroxide, is in very large quantities. In this case some zinc may be retained inthe precipitate; but, thorough washing with hot ammoniacal chloride wash water, will remove the zinc almost completely.

<u>Standard Solutionsand Standardization</u>. Potassium ferrocyanide solution was made by dissolving 21.54 grams of salt in 1 litre of distilled water.

To standardize the potassium ferrocyanide solution, 0.2 gram of pure zinc was weighed, placed in 400 c.c. beaker, and dissolved in 10 c.c. of conc. HCl and diluted with 25 c.c. of water. When all the zinc dissolved, the solution was made basic with NH_4OH, acidic with HCl, and 3 c.c. of acid added in excess.

The solution was then diluted to 200 c.c. with hot

water and heated nearly to boiling. The hot solution was divided into two portions. One portion was titrated with ferrocyanide solution, running few cubic centimeters at a time, tested on a porcelain plateto brown tinge,using uranium nitrate as outside indicator. A greater part of the second portion was then added and titrated more cautiously till end-point was again passed. Finally, the last portion was added, the beaker rinsed with a large portion of the solution, From this point the titration was completed very carefully, testing after each addition of two drops, to end-point.

The amount of ferrocyanide solution required to produce a color change under similar conditions when no zinc is present, was determined, and the correction applied.

Uranium nitrate solution was made by dissolving 1 gram of salt in 10 c.c. of water.

<u>Procedure for Zinc Determination.</u> A 0.5 gfam sample was weighed into a 400 c.c. beaker, and 10 c.c. of water, 10 c.c. HNO_3, and 3 grams of $KClO_3$ were added. In 5 minutes 10 c.c. HCl were added and the mixture heated at low temperature. To the mixture 10 c.c. of 1:1 H_2SO_4 was then added and the mixture evaporated to dryness. The residue was then cooled and taken up with 35 to 50 c.c. of ammoniacal solution (200 grams of NH_4Cl in

500 c.c. conc. NH_4OH and 750 c.c. H_2O), boiled a minute or two, bromine water added and the mixture boiled a short time to expell the excess bromine. The solution was then filtered hot and washed ten times with hot ammonium chloride mixture (100 grams NH_4Cl in 1 litre of water and 50 c.c. conc. NH_4OH).

The filtrate was neutralized with HCl and 3 c.c. of acid added in excess. Hydrogen sulphide was then passed into the filtrate to precipitate the copper. Copper was filtered out and the filtrate diluted to 200 c.c. with hot water and heated nearly to boiling. The procedure from this point to the end was similar to the titration procedure for standardization of standard potassium ferrocyanide solution.

Precious Metals Determinations.

Gold and silver in the flotation products were determined by Fire Assaying. Two different methods were employed. The "Iron Nail Method", was used on the products low in copper, such as talc froth, iron concentrates and tailings. On products high in copper, such as copper concentrates, zinc concentrates and the original ore, "Perkins' Excess Litharge Method" was used.

The Iron Nail Method. This method was used because it required no preliminary fusion for the determination of reducing power of the products, whose precious metal values were

to be determined.

This method does not attempt to oxidize the impurities, such as sulphur, arsenic and antimony, but aims to carry them into the slag. The ore is decomposed by the iron nails added to the charge and by the litharge present. The soda should be, at least, twice the amount of the ore in the charge, as this flux carries the impurities into the slag. The reactions that take place are as follows:-

$$7\ PbO + FeS_2 + 2\ Na_2CO_3 = 7\ Pb + Na_2SO_4 + FeO + 2\ CO_2.$$

Since only a limited amount of PbO is used (25 to 30 grams), the remainder of the pyrite is decomposed by iron nails, thus,-

$$FeS_2 + Fe = 2\ FeS.$$

The FeS goes into the slag.

The following type-charge was used,-

Ore	0.5 A.T.
Na_2CO_3	30 grams.
PbO	30 "
SiO_2	3 "
Borax Glass	8 "
Iron Nails	15 "
Salt cover.	

<u>Perkins' Excess Litharge Method</u>. This is the best method for determination of precious metals in products high in copper. It is based on the fact that PbO will dissolve oxides

of other metals, and if present in great excess, will prevent, to a large extent, the reduction of copper, thus preventing the latter from entering the lead button.

A preliminary assay was first made on the product whose precious metal content was to be determined.

The following type-charge was used,-

Ore	0.5 A.T.
Na_2CO_3	20 grams.
PbO	80 "
SiO_2	6 "
Borax Glass	6 "
KNO_3 (depending on the R.P. of the product).	
Salt cover.	

Procedure. The charge was fused in a gas-heated crucible furnace, poured into a mold, lead button detached and hammered into a cubic form. The lead button was cupelled in a gas-heated muffle furnace. The gold and silver alloy was weighed, parted with hot dilute nitric acid, washed with hot distilled water, dried, annealed and the gold weighed.

CHAPTER 6.

PREPARATION OF THE ORE.

COARSE CRUSHING.

In October, 1930, a shipment of 300 pounds of disseminated ore was received from the Flin Flon Mines.

Screening.

The ore was minus 2 inch size when received. It was screened over 1/4 inch and 14 mesh on a laboratory shaking screen, yielding three screen products,- plus 1/4 inch, minus 1/4 inch, plus 14 mesh, and minus 14 mesh material. Representative samples of 1/2 inch size and over were selected for microscopic and minerographic examinations.

Crushing.

All the plus 1/4 inch material was broken in a small Dodge breaker (Figure 11.), set to about 3/8 inch size. The products of the Dodge breaker were screened giving the above screen products.

All the minus 1/4 inch plus 14 mesh, from the Laboratory Shaking Screen(Figure 15.) was passed through an Allis-Chalmers sample grinder(Figure 12.) set to give about 14 mesh product. The products from the grinder was screened as before. The oversize was crushed again until all the material was minus 14 mesh.

Riffling.

When crushed to minus 14 mesh, all the material was thoroughly mixed to ensure uniformity and then riffled to give a sample of about 20 pounds. The 20-pound sample was used for preliminary chemical analyses of the ore and the screening analyses for mineral particles liberations. The balance of the material was stored in large air-tight tin container until required for experimental tests.

GRINDING.

For comparative flotation tests, it is necessary to establish a definite procedure in grinding for a given batch of ore, in order to eliminate one variable from flotation procedure.

Procedure for Screening.

Since 1500 gram batches were to be used in the flotation cell, the same amounts had to be taken for the grinding and the screening tests.

Screen analyses were carried out by screening a portion of the 1500 gram. batch in Tyler Standard Screens. The sieves were shaken in Rotap Apparatus (Figure 16.) from ten to twenty minutes. The length of time for shaking and the size of the portion, depended on the fineness of material to be screened. Fine

material required a longer period of shaking and small amounts were taken.

Mineral Liberation Due to Grinding.

Fine degree of grinding in flotation practice is to unlock the minerals so that no one particle, which is required to be separately recovered, may be a composite of more than one mineral.

A test was made to determine the effect of grinding on the various minerals present in the ore. The procedure was as follows,- a unit batch of ore was dry ground with 3020 grams of balls in the Denver Laboratory Weinig type ball-mill for 15 minutes. Screen analysis was made of the ground product and the metal values in each screen size determined. The results are reported in Table No. 8.

The metal values in various screen sizes, of the grinding test, showed that the zinc blende and the chalcopyrite were readily ground to minus 200 mesh. After the ore was ground, the per cent of the zinc blende retained on various screen sizes was slightly higher than the per cent of the chalcopyrite. This would be expected since the hardness of the zinc blende, which is the iron variety, is slightly higher than the hardness of the chalcopyrite. In general, the percentages of the zinc blende and the chalcopyrite retained on various screen sizes seem to

run very close together. This seems to indicate that the two minerals were still locked together. This would be expected since the two minerals, which appear as solid solutions and have solidified practically at the same time from magmatic solutions, may still be composite particles. Hence high degree of grinding would be necessary to liberate the mineral particles.

The mineralogical analyses showed that the pyrite was readily crushed to minus 100 plus 200 mesh and with greater difficultythan the zinc blende and the chalcopyrite beyond 200 mesh. This was probably due to the fact that the average size of the pyrite crystals in the ore was over 200 mesh. It may be probable that the pyrite minerals, to plus 200 mesh size, were only liberated from the other minerals and remained as whole crystals, retaining their original pyrite form. Grinding beyond 200 mesh size would be crushing each individual pyrite crystal to smaller size.

The per cent of the pyrite retained on the various screen sizes to plus 100 mesh ran close to that of the zinc blende and the chalcopyrite, but below 100 mesh size, the amount of the pyrite retained was greater. This would be expected since pyrite occurs as individual crystals in the ore and is practically not interlocked with the zinc blende and the chal-

copyrite . It is probable that the precious metals are associated with the zinc blende and the chalcopyrite. The Ottawa Laboratories have come to the following conclusion,- "No mineral was detected that would indicate the presence of gold and silver. It is concluded, therefore, that these precious minerals occur in a state of extreme subdivision and will probably defy concentration. They apparently occur associated with either the chalcopyrite, the sphalerite or the pyrite (25)."

In conclusion, it may be said that grinding to minus 200 mesh is necessary to unlock the mineral particles in the Flin Flon disseminated ore. This degree of grinding fairly well liberates the mineral particles for flotation purpose.

Grinding Tests.

A series of tests on ball-mill dry grinding was carried out, to determine the length of time and the weight of balls required to a unit batch which would give the desired degree of fineness of grinding. The results obtained are shown in Table No. 9.

The two-hour grind with 4520 grams of balls gave the maximum amount of fine material, but the product on screen-

ing, tended to blind the 200 mesh screen, thus preventing complete screening.

The one-hour grind with 6010 grams of balls gave a product with over 99 per cent minus 100 mesh and practically 90 per cent minus 200 mesh. This material screened readily and as it was practically equivalent to the accepted practice at the Flin Flon mill today, this grind was established as the standard for the flotation tests.

TABLE NO.8. METAL VALUES OF VARIOUS SCREEN SIZES.

Mesh	Wgt %	Chemical Analysis. Insol %	Fe-%	Cu-%	Zn-%	An-Oz.	Ag-Oz.	Mineralogical Analysis Per Cent. Gangue Min.	Pyrite	Chalcopyrite	Zinc Blende	Per Cent Retained. Gangue Min.	Pyrite	Chalcopyrite	Zinc Blende	Au-	Ag-
Before Grinding.																	
Heads		25.72	24.74	1.90	5.09	.10	1.32	37.0	47.9	5.48	9.60						
-14 + 28	32.5	25.20	25.47	1.80	4.87	.08	.94	35.9	49.7	5.19	9.19	31.5	33.8	30.7	31.1	26.0	23.2
-28 + 48	15.9	26.06	24.56	1.95	5.12	.10	1.50	37.2	47.5	5.62	9.66	16.0	15.8	16.2	16.1	15.9	18.1
-48 +100	10.8	26.72	24.36	1.73	4.83	.10	1.50	38.4	47.5	4.99	9.11	11.2	10.7	9.8	10.2	10.8	12.2
-100+200	10.3	24.10	27.70	1.55	4.29	.10	1.49	32.4	55.0	4.47	8.10	9.0	11.8	8.4	8.7	10.3	11.7
-200	30.5	26.14	24.5	2.17	5.64	.12	1.62	36.0	47.1	6.26	10.64	29.7	30.1	34.9	33.8	36.6	34.7
After Grinding.																	
-14 + 28	3.4	37.26	18.75	.90	2.88	.06	.67	51.0	41.0	2.60	5.43	4.7	2.9	1.5	1.9	2.0	1.7
-28 + 48	12.2	34.40	21.90	1.48	4.22	.08	1.03	45.0	42.8	4.27	7.96	14.8	10.9	9.5	10.1	9.8	9.4
-48+ 100	19.8	31.00	23.42	1.59	4.55	.09	1.12	41.1	45.7	4.59	8.59	22.0	18.9	16.6	17.7	17.8	16.8
-100+200	17.2	22.50	28.49	1.68	4.89	.10	1.22	29.4	56.4	4.84	9.33	13.7	20.2	15.3	16.7	17.3	16.0
-200	47.4	22.96	25.05	2.26	5.77	.12	1.56	34.9	47.7	6.53	10.87	44.6	46.9	56.4	53.7	55.9	56.0

Zinc blende calculated on basis of 53% Zn, 8% Fe, 39% S.

TABLE NO. 9. SCREEN ANALYSES OF GRINDING TESTS.

Mesh.	Per Cent Retained.			
	Before Grinding	Ground 1 hr. 4520 grams of balls.	Ground 2 hrs 4520 grams of balls.	Ground 1 hr. 6010 grams of balls.
- 14 + 28	32.5	0.4	0.3	0.1
- 28 + 48	15.9	0.4	0.3	0.2
-48 + 100	10.8	0.7	0.1	0.5
-100 + 200	10.3	15.2	5.1	10.3
-200	30.5	83.3	93.7	89.0

FIG. 15 SHAKING SCREENS

FIG. 16 TYLER SCREENS AND ROTAP APPARATUS

CHAPTER I.

SELECTIVE FLOTATION.

Experimental work in flotation, for any ore, is principally tests on the flotation reagents and the amounts required to obtain the best results. In the present investigation on the flotation of the Flin Flon disseminated ore, the reagents selected were, with few exceptions, those adopted at the Flin Flon 25-ton Pilot mill. Pine oil was used instead of aerofloat and cresylic acid. The amounts of the reagents, to be added, were based on those used in the 25-ton Pilot mill. The reagents were added to the pulp in solid form.

STANDARDIZATION OF TEST PROCEDURE.

Before any experimental work could be undertaken, it was necessary to determine the length of time required for agitation and for removal of froth in the various stages of the operation.

Preliminary Test No. 1.

The Laboratory type Mineral Separation Sub-A machine was partly filled with water. The impeller of the machine was started and 1500 grams of ore were placed in the machine. Water was then added until the level of the pulp was slightly above the level of the partition between the agitation compartment and

the froth separation compartment. This gave a dilution of about 3 to 1. The pulp was agitated in the flotation cell for 30 minutes with 2.00 pounds of lime, and 2.00 pounds of zinc sulphate to the ton of dry weight of ore charged. At the end of 30 minutes conditioning, four drops of pine oil were added, air admitted and the pulp floated for talc; collecting three talc froths, each of five minutes intervals.

To float copper, 2.8 pounds of NaCl were added and the pulp agitated five minutes; followed by 0.04 pound of potassium ethyl xanthate, and one minute agitation, and then 2 drops of pine oil, collecting two, five minutes interval, froths.

For zinc flotation, 2.0 pounds of lime, 0.74 pound of copper sulphate were added, agitated for 10 minutes, followed by 0.12 pound of xanthate and 2 drops of pine oil, and two, five minutes interval, froths collected.

Iron was floated with 0.3 pound of xanthate and 8 drops of pine oil, collecting the froth for 10 minutes.

The machine was then drained of the remaining product, thoroughly washed, and filled with water. The flotation products were placed on a steam heated drier to dry. The tailings were dewatered by decantation and likewise dried.

Table No. 10. gives the amounts of each product obtained during each five-minutes interval.

TABLE NO. 10. PRELIMINARY TEST, NO. 1.

Period Min.	Talc Froth -%-	Cu Froth -%	Zn Froth -%	Fe Froth -%	Tails -%
0 - 5	4.24	14.10	22.60	12.65	
5 -10	1.98	1.50	3.35	2.80	
10 -15	1.78				
					35.10

Examination of each of these products showed that the copper froth, the zinc froth, and the iron froth collected during the second five-minute interval were in all cases largely composed of gangue minerals;and as the total amount collected in each case was relatively small,it was clearly indicated that a five-minute collection of froth for each flotation was sufficient. In excess of that time period the concentrates were too greatly diluted with gangue minerals. The talc froth showed a gradual increase in metallic sulphides with increasing time of froth removal, indicated by a darkening of the froth. However, as there was also a steady increase in removal of this objectionable impurity, it was decided to standardize on a talc froth removal of from 10 to 15 minutes, to ensure complete removal of the talc;although it

was expected that this longer period would tend to give metallic sulphide losses in the talc froth.

Preliminary Test No. 2.

The zinc concentrate produced in this first experiment showed an excessive iron content. It was assumed this represented insufficient pyrite depression, due to insufficient conditioning prior to zinc flotation. To check this point a second experimental run was made increasing the time of zinc conditioning from 10 minutes to 30 minutes. Reagents used were the same as in Test No. 1. with the following exceptions,- 30 minutes conditioning prior to zinc flotation, 6 drops instead of 8 drops of pine oil added to the cell for iron flotation, and the froths from the copper, the zinc and the iron flotations were collected for five minutes. Table No. 11. shows the results obtained,-

TABLE NO.11. TEST NO.2.

Products	Per Cent Wgt.	Per Cent Analysis				Per Cent Recovery or Loss			
		Insol	Fe	Cu	Zn	Insol	Fe	Cu	Zn
Talc Froth	10.15	49.94	11.97	2.46	5.08	19.70	4.93	13.14	10.14
Cu Conc.	15.70	8.12	24.81	6.62	17.25	4.96	13.12	54.60	53.25
Zn "	37.30	4.84	42.76	1.45	4.32	7.02	64.50	28.50	31.70
Fe "	4.87	15.22	34.58	.70	1.44	2.89	6.81	1.79	1.38
Tailings	31.90	53.30	8.92	.11	.53	66.10	11.50	1.83	3.33
Heads		25.72	24.74	1.90	5.09				

The results obtained (Table No. 11) showed no improvement due to increased time of conditioning prior to zinc flotation; and therefore it was considered that a 10 minute conditioning period prior to zinc flotation was sufficient.

General Procedure of Tests for Flotation.

From the standardization tests results, the following procedure was established for routine experimental flotation tests,- A unit batch of the Flin Flon disseminated ore was dry ground in the ball-mill with 6010 grams of balls for one hour, yielding a product of approximately 99 per cent minus 200 mesh (See Chapter 6). The ore was placed with lime and zinc sulphate in the Laboratory Sub-A flotation machine, and the pulp conditioned for 30 minutes. Talc was removed for 12 minutes with pine oil as frothing reagent. To float copper, the pulp was conditioned for ten minutes. with salt; collecting and frothing agents were then added and copper froth collected for five minutes. To float zinc, the pulp was conditioned for ten minutes with zinc conditioning and activating reagents; collecting and frothing agents were then added and zinc collected for five minutes. To the zinc tailing, collecting and frothing reagents were added and iron froth collected for five minutes. The machine was then drained, thoroughly washed, and filled with water. The flotation products were then dried; the tailings dewatered and dried.

The results obtained by this procedure, as shown in the results of Test No. 2. (Table No.11.) were far from satisfactory. The copper concentrates carried as much of the zinc (53.25%) as of the copper (54.6%) and the grade of copper concentrates (6.62%) was low. Also the zinc concentrates, although collecting practically 90% of the remaining zinc (31.7% out of a balance of 36.4%) and a similar amount of the remaining copper(28.5% out of a balance of 32%) was likewise a low grade concentrate (4.32% zinc). However, the losses of copper (1.83%) and of the zinc (3.33%) were reasonably low, and it was, therefore, considered that this would prove a satisfactory starting point for further work.

SELECTIVE FLOTATION TESTS.

Having established a procedure which gave low losses in copper and zinc to the tailings, and at the same time fixed such variables as time periods of conditioning and of frothing, pulp density, degree of fineness of grinding, etc., the next step was to attempt to adjust reagent amounts to improve the selectivity of the different floats so as to produce a maximum selection of the desired minerals with minimum contamination of a given mineral concentrate with minerals from preceding or succeeding operations. The amount of ore available did not

allow of multiple tests under identical conditions to produce a given concentrate in sufficient quantities to allow cleaning or recleaning. Therefore, it had to be accepted that the quality of the concentrate produced would not equal or even be directly comparable with the results of the 25-ton pilot mill. It was assumed that satisfactory selective flotation would be realized if it were possible to obtain concentrates, which obviously could be cleaned satisfactorily by subsequent operations, and at the same time did not indicate excessive costly reagent consumption.

It has been well established by practice that the best procedure for selective flotation of copper-zinc-iron ores is to first float the copper then the zinc, and lastly the iron. This means that the copper concentrate first produced should carry a maximum amount of that metal. It may also carry considerable amounts of zinc and of iron, as these metals can be removed in cleaning operations on the copper concentrates, and returned to the copper circuit without seriously interfering with subsequent operations. Similarly the zinc concentrate may contain appreciable quantities of iron. But it should not carry any great amount of copper, as this cannot be recovered without separate retreatment circuits. The copper may be re-

moved from the zinc circuit in the cleaning operations by suitable depressants, but the copper product contained cannot go back to the copper circuit, as it will carry copper depressing agents with it, which will tend to reduce the copper recovery in that circuit.

With these facts in mind, it is obvious that selective flotation experiments, where no attempt is made to clean the concentrates, would show satisfactory results when the copper concentrate shows a high recovery of the total copper present, even if the concentrate also carries considerable zinc. On the other hand, the zinc concentrate should be low in copper, as presumably all copper has been taken off prior to zinc float. Both of these concentrates may carry considerable quantities of iron, as this metal being the last to be floated, can be returned to either circuit without doing harm. Because both chalcopyrite and marmatite carry iron in their chemical composition, there is certain to be iron in both concentrates in any event.

COPPER FLOTATION TESTS.

Accepting these assumptions and the results of Preliminary Test No. 2. (Table No. 11.) as indicating satisfactory

control of tailings; tests were run to endeavor to modify the reagent additions in such a way that the zinc and iron content of the copper concentrates would be reduced and at the same time make a greater total recovery of the copper in these concentrates.

There was three possible variables to be considered, pulp alkalinity, as effected by change in quantity of lime added to the circuit, greater depression of zinc by greater amount of zinc sulphate, and variations in kind and quantity of copper activators. As lime is an active depressent for iron, changes in pulp alkalinity should be directly reflected on the iron in the concentrates.

The results of the tests for copper recovery are shown in Table No. 12. (see addition page 104)

__Discussion of Copper Flotation Results.__

__Effect of Lime.__- With same reagents and same amounts as were used at the 25-ton Pilot mill, the pH of the pulp in the copper circuit was below 7 (Table No. 12. Test No. 1.), which meant that the pulp was either neutral or acid. Since the present flotation practice is to use alkaline pulp, a series of tests Nos. 2, 3, 4, 5, 6 and 7 was run in order to raise the alkalinity in the copper circuit by increasing the

lime in conditioning circuit. The results of these tests (Table No. 12) showed that almost six pounds of lime were required to raise the alkalinity to that used at the 25-Ton Pilot mill (pH of 9.4); and over six pounds of lime, to raise it to that used at the Flin Flon Concentrator at the present day (pH of 10. to 11.).

The results of these tests (Tests Nos. 1,2,3,4,5, 6 and 7) showed that the highest copper recovery (57.5%) was obtained ~~was~~ when 2.00 pounds of lime were used (Test No. 1.); but in this test, an appreciable amount of pyrite floated with the copper float. The best results were obtained, when the pH of the copper circuit was 9.7 (Test No.5.); 53.7 per cent of the copper (in the feed to the copper cells) was recovered, analyzing 10.90 per cent copper in the copper concentrate. The copper loss in this test was 40.7 per cent. The only iron present was practically that which is carried in the chemical composition of the chalcopyrite and the zinc blende.

Increasing the lime increased the zinc in the copper concentrates. It is evident that increasing the lime increases the depressing effect on the chalcopyrite and hence the xanthate acts on the zinc blende in preference to the copper minerals. The zinc in the copper concentrates would be removed in the copper cleaning operations.

<u>Effect of Soda Ash</u>. Soda ash used in place of lime (Tests Nos. 8 and 9) yielded very low results; only 38.3 per cent copper recovery was obtained; The copper concentrates assayed 29.61 per cent iron, which is far too highto be the iron carried in the chalcopyrite and zinc blende, showing that appreciable amount of pyrite was floated with the copper float. An appreciable amount of copper (25.8 per cent) floated with the talc froth. Soda ash, without a depressing agent for pyrite and chalcopyrite, will not yield satisfactory results, when used as a conditioning agent for separation and concentration of the Flin Flon ore by selective flotation process.

<u>Effect of Zinc Sulphate.</u> An appreciable amount of zinc floated with the copper float. An attempt was made to decrease the zinc in the copper float by increasing the zinc sulphate (a depressant for zinc blende). The results of the tests Nos. 10, 11, 12 and 13 (Table No. 12) showed that increasing the zinc sulphate did not, appreciably decrease the zinc in the copper concentrate. However, increase of zinc sulphate increased the copper recoveries about 3 per cent and decreased the copper losses by approximately 2 per cent, also

increased the copper in the copper concentrate from about 10.9 per cent to about 11.3 per cent.

Increasing zinc sulphate caused the alkalinity of the circuit pulp to increase, this is due to hydrolysis of that salt.

<u>Effect of Thiocarbanilid</u>. An attempt was made to use thiocarbanilid in place of potassium ethyl xanthate Tests Nos. 14 and 15). The copper recoveries were very low (22.9% in Test No. 14. and 30.8% in Test No. 15.). Evidently, thiocarbanilid is not as powerful as collector as xanthate.

<u>Effect of Sodium Sulphide and Sodium Cyanide.</u> During the conditioning operation, there is a possibility that a little of soluble copper salt is produced which tends to promote the flotation of zinc. C.S. Parsons (25) suggested that probably sodium sulphide and a little of cyanide may help to get rid of this copper. Test No. 16. was run using sodium sulphide and cyanide. The results of the tests showed lower copper recovery (43.5%), lower grade of copper concentrate (9.26% copper) and higher zinc in the copper concentrates (28.29% zinc) than without the sodium sulphide and cyanide. This is probably due to that fact that sodium sulphide has an inhibiting effect on

the chalcopyrite; and a sulphidizing effect on the zinc blende, which tends to make the zinc minerals more readily floatable.

ZINC FLOTATION TESTS.

The results of the Preliminary Test No. 2. showed that a satisfactory zinc concentrate could be obtained. Tests were run to endeavor to modify the reagents in such a way that copper and the iron content of the zinc concentrate would be reduced, and at the same time, make a greater recovery of the zinc in the concentrate.

The results of the tests for zinc recovery are shown in Table No. 13.

Discussion of Zinc Flotation Results.

The zinc in the talc froth and the copper concentrates would be recovered in the cleaning operations of those concentrates. Only the zinc concentrates will be discussed here.

In the Preliminary Test No. 2. an appreciable amount of iron (64.50%) floated inthe zinc float (Table No. 11.). It was thought, that probably the xanthate used (0.12 pound in Preliminary Test No. 2.) was in excess of the amount required to collect the zinc minerals; the excess xanthate acting on the pyrite, thus floating the iron minerals in the zinc float.

In Test No. 1., the amount of xanthate was reduced to 0.08 pound per ton of ore. A decrease of xanthate decreased the iron in the zinc concentrate (analysis,- 29.12% in Test No.1. as against 42.76% in Preliminary Test No.2.). It was considered that the iron content in the zinc concentrates was still too high, and could be reduced by a suitable depressing agent.

Effect of Lime. The results of the tests (Table No. 13.) showed that any amount above 5.00 pounds of lime in the conditioning circuit and 2.00 pounds in the zinc circuit (Test No. 10.) was sufficient to depress practically all the pyrite and keep it out of the zinc froth. Any amount, over 5.00 pounds of lime in the conditioning circuit, would be a waste of that reagent. Furthermore, the results showed that lime used in excess of 5.00 pounds in the conditioning circuit and 2.00 pounds in the zinc circuit decreased the zinc recovery.

Effect of Soda Ash. In addition to what has been said previously with respect to this reagent, it may be said that soda ash yielded a higher recovery of zinc than the lime, (92.3% with soda ash; 88.0% with lime). Although the zinc recovery was higher, the results were not satisfactory, as the zinc concentrates carried an appreciable amount of iron.

__Effect of Zinc Sulphate.__ Zinc sulphate is the practical depressant for zinc blende. The effect of this reagent upon the zinc blende has been discussed in the "Discussion of Copper Flotation Results". It may be added here, that any amount in excess of 2.67 pounds of zinc sulphate decreased the zinc recoveries to less than 88.0 per cent (Tests Nos. 10, 11, 12 and 13, Table No. 13.) Although the results of Test No. 11. show a slight increase of zinc recovery with the increase of zinc sulphate, but this may be due to the collecting action of the cresol (effect of cresol discussed on Page 103).

__Effect of Thiocarbanilid.__ Thiocarbanilid, used in place of xanthate, yielded very low zinc recovery (1.6%), as well as a very low grade zinc concentrate (7.57 per cent zinc).

__Effect of Sodium Sulphide and Cyanide.__ Sodium sulphide and cyanide yielded a zinc recovery of 79.5 per cent (Test No. 16.), but the grade of zinc was low (16.18% zinc).

__Effect of Cresol.__ In Tests Nos. 11, 12, 13 and 16,

cresol was used in place of pine oil, in the zinc circuit. The results of the tests showed that the cresol had a slight collecting action on the zinc blende, but it also had a collecting action on the pyrite, thus raising the iron in the zinc concentrate.(25.79% iron in Test No. 16; 17.34% iron in Test No. 10.)

IRON FLOTATION TESTS.

In the present investigation iron was also floated. The results of the tests are shown in Table No. 14.

Discussion of Iron Flotation Results.

The results of the copper and the zinc flotation tests showed that the pyrite, with proper depressing reagents and sufficient amounts of those reagents, could be practically excluded from the copper and the zinc concentrates.

Effect of Lime. Test No. 1. showed that 2.00 pounds of lime in the conditioning circuit and 2.00 pounds of lime in the zinc circuit was not sufficient to depress the pyrite from floating with the copper and the zinc froths. Since lime has a depressing effect onthe pyrite, it is evident that increase of lime will decrease the iron recovery. The flotation

test results showed this to be the case.(Table No. 14.)

Effect of Soda Ash. Tests, with soda ash used in place of lime, showed that a satisfactory iron concentrate could not be obtained (Test No. 9.) since the pyrite floated with the copper and the zinc froths. Soda ash has little, if any, depressing effect on the pyrite.

Effect of Zinc Sulphate. Zinc sulphate has no effect on the pyrite, or the recovery of the iron, but increasing the zinc sulphate increased the zinc in the iron concentrates since the zinc in the zonc concentrates was decreased. (See "Discussion of Zinc Flotation Results").

Effect of Thiocarbanilid. The properties of thiocarbanilid, as a collecting agent has been discussed previously. It also yielded low iron recoveries in the iron concentrates (10.6%).

Effect of Sodium Sulphide and Sodium Cyanide. Sodium sulphide and cyanide have a depressing effect on the pyrite. These reagents added, along with lime, to the conditioning circuit yielded an iron recovery of 67.8 per cent Test No. 16.). Approximately the same amount of reagents in Test No. 6. yield-

ed an iron recovery of 64.2 per cent. But in Test No. 6. the grade of iron concentrates was higher (43.16% iron in Test No. 6; 39.52% iron in Test No. 16.).

Effect of Xanthate. In some flotation tests, the amount of xanthate used in the iron circuit was varied. The results showed that 0.20 pound of xanthate was sufficient to collect the pyrite in the zinc tailings.

Recovery of Gold and Silver.

Precious metal values were determined in the flotation products in some of the tests. The assays (Table No.15) showed that the silver was concentrated mostly in the copper and the zinc concentrates. The gold was mostly concentrated in the zinc concentrate. Over 85 per cent of the gold and over 75 per cent of the silver floated with the sulphide minerals froths. From 6 to 7 per cent of the gold and from 11 to 13 per cent of the silver floated with the talc froth. Appreciable amount of the precious metals in the talc froth would be recovered in the talc cleaning operations. From these figures (Table No. 15.) it is evident that a large percentage of the precious metals would be floated with the sulphide concentrates and only a small percentage would remain

in the tailings. In such a case, the precious metal values in the tailings could be accepted as a loss, as the cost of the cyanidation treatment would not justify their recovery.

SUMMARY AND CONCLUSIONS.

The results of the trests on the selective flotation may be summarized as follows,-

1. At least 6 pounds of lime was required in the conditioning circuit to maintain an alkalinityin the copper circuit equivalent to that which is practiced at the Flin Flon Concentrator.
2. Increase of lime in the conditioning circuit decreased the copper recovery, and increased the zinc in the copper concentrates, since the xanthate acted on the zinc blende in preference to the depressed copper minerals.
3. Soda ash yielded very low copper, and iron recoveries. The zinc recovery was high, but the zinc concentrates carried an appreciable amount of iron.This reagent cannot be used to yield satisfactory results without a depressing agent for pyrite and chalcopyrite.
4. Thiocarbanilid yielded low recoveries. It is not as powerful a collector as the xanthate. Since thiocarbanilid is not very soluble in water and has to be

dissolved in toduidin (a very expensive reagent), the cost of the toluidin would justify the use of thiocarbanilid as a collector in place of xanthate.

5. Sodium sulphide and sodium cyanide produced no decrease of zinc in the copper concentrate, in fact, opposite results were obtained, the zinc was activated. These reagents showed a depressing effect on the chalcopyrite and pyrite.

6. Cresol showed a slight collective action on the zinc blende and the pyrite, thus increasing the iron in the zinc concentrates.

7. Over 85 per cent of the gold and over 75 per cent of the silver floated with the copper, the zinc and the iron concentrates.

8. The most satisfactory recoveries and grades of concentrates were obtained when the following reagents and amounts were used,-

To Conditioning Circuit:-		
Lime	5.00	pounds
Zinc Sulphate	2.67	"
To Talc Circuit:-		
Pine Oil	.24	"
To Copper Circuit:-		
NaCl	2.80	"
Xanthate	.04	"
Pine Oil	.12	"
To Zinc Circuit:-		
Lime	2.00	"
Copper Sulphate	0.75	"
Xanthate	.08	"

	Pine Oil	.12 pounds
To	Iron Circuit:-	
	Xanthate	.20 "
	Pine Oil	.36 "

These reagents gave recoveries of 55.0 per cent of the copper and 88.0 per cent of the zinc.

COMPARISON OF RESULTS.

A compairson between the reagent consumptions, grade of concentrates and recoveries of metals obtained in the present investigation and those of the 25-Ton Pilot mill is shown in Tables Nos. 16 and 17.

Reagent Consumption.

Total reagent consumption is appreciably higher in the present investigations, but this is almost entirely made up of additional lime to the conditioning circuits. It has been pointed out elsewhere that it was necessary to greatly increase the lime supply to the flotation circuit to obtain anywhere near the pulp pH value that was used at the Pilot mill, and it is quite probable that difference in mineral content of the waters used would account for the difference.

Cleaning the rougher concentrates in the present investigation would have increased the reagent consumption.

Presumably, the amounts used for copper recovery would have been higher; and those used for zinc recovery would, probably, have been equal to the amounts used at the Pilot mill.

Recoveries.

The copper recoveries were not as high as those obtained at the 25-Ton Pilot mill; the zinc recoveries were slightly higher. The grade of the concentrate was not as high. It is possible that cleaning concentrates would raise the metal values, as well as, decrease the impurities.

The precious metals distributions in the flotation products obtained by the writer and the products obtained at the Pilot mill, vary in some respects. In the present tests, the gold was concentrated mostly in the zinc concentrate; while the Pilot mill results showed that the gold was concentrated mostly in the copper concentrates. The gold and silver recoveries in the copper and the zinc concentrates were practically the same in both, the Pilot mill tests and the present investigation tests.

Addition.

Recovery percentages of various flotation products were calculated on the basis of the metal content in the feed to the respective circuits; loss percentages were based on the tailings of the same (respective) circuits.

TABLE NO. 12. SELECTIVE FLOTATION TESTS FOR COPPER.

Test No.	Reagents - Pounds Per Ton of Ore										pH	Products - Percentages						
	Conditioning					Talc	Copper					Name	Wgt.	Analysis			Recov'	Losses
	CaO	Soda Ash	Na_2S	NaCN	$ZnSO_4$ $7H_2O$	Pine Oil	NaCl	KEtX	Thio	Pine Oil				Fe	Cu	Zn	Cu	Cu
												Heads		24.74	1.90	5.09		
1	2.00				2.00	.24	2.80	.04		.12	<7	Talc Fr	9.77	11.64	2.41	4.91	12.5	
												Cu Conc	14.60	25.93	6.69	19.33	57.5	35.7
2	4.00				2.00	.24	2.80	.04		.12	<7	Talc Fr	8.09	12.06	2.93	5.99	12.5	
												Cu Conc	8.23	14.65	10.53	15.83	51.3	42.7
3	4.67				2.00	.24	2.80	.04		.12	<7	Talc Fr	8.04	11.23	2.81	5.99	11.9	
												Cu Conc	8.04	14.04	10.50	18.21	50.5	43.4
4	5.33				2.00	.24	2.80	.04		.12	7.3	Talc Fr	7.74	11.22	3.01	4.53	12.3	
												Cu Conc	8.0[illegible]	16.90	10.86	21.98	52.8	41.4
5	6.00				2.00	.24	2.80	.04		.12	9.7	Talc Fr	7.78	11.78	2.92	4.27	12.0	
												Cu Conc	8.26	17.04	1090	21.63	53.7	40.7
6	6.67				2.00	.24	2.80	.04		.12	11.3	Talc Fr	7.74	12.60	3,04	4.47	11.4	
												Cu Conc	8.38	18.36	9.64	23.09	47.6	46.4
7	8.00				2.00	.24	2.80	.04		.12	11.2	Talc Fr	7.23	16.69	2.71	4.89	10.3	
												Cu Conc	8.45	18.90	8.68	26.46	43.1	50.9
8		4.00			3.33	.24	2.80	.04		.12	<7	Talc Fr	Unsuccessful					
												Cu Conc	mostly pyrite.					
9		6.00			4.00	.24	2.80	.04		.12	7.9	Talc Fr	14.17	15.22	3.45	4.29	25.8	
												Cu Conc	8.72	29.61	6.22	14.80	38.3	45.5
10	5.00				2.67	.24	2.80	.04		.12	8.7	Talc Fr	8.26	12.44	2.93	6.00	12.8	
												Cu Conc	8.05	16.34	11.34	20.19	55.0	39.2
11	5.00				4.00	.24	2.80	.04		.12	8.7	Talc Fr	7.90	12.48	2.95	5.51	12.3	
												Cu Conc	8.22	16.22	11.44	19.32	56.0	38.6
12	6.00				5.00	.24	2.80	.04		.12	11.2	Talc Fr	8.10	12.17	2.83	5.40	12.1	
												Cu Conc	8.28	16.22	11.35	21.42	56.4	38.6
13	6.00				6.00	.24	2.80	.04		.12	11.3	Talc Fr	7.84	12.89	2.80	5.35	11.6	
												Cu Conc	8.66	16.92	11.12	20.18	57.5	37.4
14	[illegible].00				2.67	.24	2.80	.04	.04	.12	8.5	Talc Fr	8.54	12.33	2.84	5.57	12.8	
												Cu Conc	3.43	15.44	13.03	11.20	22.9	67.1
15	5.00				3.33	.24	2.80	.04	.08	.12	8.7	Talc Fr	8.25	17.04	2.86	5.67	12.4	
												Cu Conc	4.75	16.83	10.81	17.19	30.8	60.0
16	6.00		2.00	.02	2.00	.24	2.80	.04		.12	11.4	Talc Fr	7.68	12.71	2.67	4.80	11.8	
												Cu Conc	7.82	14.00	9.2[illegible]	28.29	43.5	50.0

TABLE NO. 13. SELECTIVE FLOTATION TESTS FOR ZINC.

Test No.	Reagents - Pounds Per Ton of Ore*										pH	Products - Percentages						
	Conditioning					Zinc Flotation Cells						Name	Wgt.	Analysis			Recov.	Loss.
	CaO	Soda Ash	Na_2S	NaCN	$ZnSO_4 \cdot 7H_2O$	CaO	$CuSO_4 \cdot 5H_2O$	KEtX	P.O	Cresol				Fe	Cu	Zn	Zn	Zn
												Heads		24.74	1.90	5.09		
1	2.00				2.00	2.00	0.75	.08	.12		<7	Zn Conc.	15.15	29.12	3.08	8.95	75.9	8.5
2	4.00				"	"	"	"	"		11.3	Zn Conc.	15.70	30.29	3.68	15.70	74.7	16.4
3	4.67				"	"	"	"	"		11.5	Zn Conc.	12.17	29.85	5.19	20.88	82.2	11.0
4	5.33				"	3.33	"	"	"		11.3	Zn Conc.	9.47	19.11	4.11	23.61	75.2	14.4
5	6.00				"	"	"	"	"		11.5	Zn Conc.	9.50	17.45	4.44	24.29	77.9	13.0
6	6.67				"	"	"	"	"		11.4	Zn Conc.	8.65	22.26	5.16	23.33	72.0	15.5
7	8.00				"	"	"	"	"		11.4	Zn Conc.	7.64	23.82	6.56	23.19	71.0	14.3
8		4.00			3.33	Soda Ash 2.00	"	"	"			Zn Conc.	Mostly pyrite.					
9		6.00			4.00	2.00	"	"	"		8.3	Zn Conc.	41.30	39.75	1.78	7.11	92.3	4.9
10	5.00				2.67	CaO 2.00	"	"	"		11.4	Zn Conc.	10.44	17.34	5.35	25.01	88.0	7.1
11	5.00				4.00	"	"	"		.12	11.5	Zn Conc.	14.72	24.18	3.54	18.38	88.5	7.1
12	6.00				5.00	"	"	"		"	11.8	Zn Conc.	12.20	24.48	4.33	20.21	86.6	7.5
13	6.00				6.00	"	"	"		"	11.8	Zn Conc.	9.98	22.88	5.19	23.38	77.8	11.5
14	5.00				2.67	"	"	Thio .08		"	11.4	Zn Conc.	.89	25.86	2.48	7.57	1.6	81.6
15	6.00		2.00	.02	2.00	"	"	KEtX .08		"	11.5	Zn Conc.	12.31	25.79	5.67	16.18	79.5	10.1

TABLE NO. 14. SELECTIVE FLOTATION TESTS FOR IRON.

TEST No.	Reagents Pounds Per Ton of Ore — Conditioning: CaO	Conditioning: Na_2CO_3	Conditioning: Na_2S	Conditioning: NaCN	Conditioning: $ZnSO_4$ $7H_2O$	Zinc: CaO	Zinc: Nada Ash	Iron: KEtX	Iron: Thio	Iron: Pine Oil	Products – Percentages: Name	Wgt.	Analysis: Fe	Analysis: Cu	Analysis: Zn	Recov Fe	Loss Fe
1	2.00				2.00	2.00		.20		.36	Fe Conc.	28.80	40.86	.56	.80	80.5	
											Tailings	31.50	11.24	.55	.55		14.2
2	4.00				"	"		"		"	Fe Conc.	30.50	45.74	.41	1.28	78.5	
											Tailings	37.40	11.43	.18	.59		15.6
3	4.67				"	"		"		"	Fe Conc.	33.22	44.13	.37	1.37	77.2	
											Tailings	38.50	12.44	.18	.48		17.7
4	5.33				"	3.33		"		"	Fe Conc.	39.90	36.37	.83	1.26	69.7	
											Tailings	34.84	18.60	.28	.71		25.0
5	6.00				"	"		"		"	Fe Conc.	34.00	40.30	.77	1.23	66.2	
											Tailings	40.40	17.15	.29	.61		28.3
6	6.67				"	"		"		"	Fe Conc.	22.50	43.84	1.39	2.15	64.2	
											Tailings	52.80	19.80	.26	.53		28.9
7	8.00				"	"		"		"	Fe Conc.	22.46	43.16	1.46	1.76	48.0	
											Tailings	54.22	20.13	.26	.59		42.4
8		4.00			3.33		2.00	.267		"	Fe Conc.	Unsuccessful, mostly gangue.					
											Tailings						
9		6.00			4.00		"	.200		"	Fe Conc.	25.18	22.74	.88	2.17	34.2	
											Tailings	30.60	9.33	.21	.46		9.4
10	5.00				2.67	2.00		.20		"	Fe Conc.	36.35	43.12	.42	.53	76.5	
											Tailings	36.90	13.64	.17	.42		19.4
11	5.00				4.00	"		"		"	Fe Conc.	32.20	43.70	.55	.81	74.5	
											Tailings	37.00	13.00	.15	.41		19.1
12	6.00				5.00	"		"		"	Fe Conc.	27.80	44.11	.30	1.07	63.2	
											Tailings	43.60	17.07	.17	.41		28.8
13	6.00				6.00	"		.133		"	Fe Conc.	21.85	41.32	.38	1.87	45.0	
											Tailings	51.70	21.07	.15	.41		44.4
14	5.00				2.67	"			.30	"	Fe Conc.	7.75	34.10	3.49	21.50	10.6	
											Tailings	79.40	25.41	1.13	3.16		82.7
16	6.00		2.00	.20	2.00	"		.20		"	Fe Conc.	33.60	39.52	.48	1.22	67.8	
											Tailings	38.60	15.63	.18	.41		25.3

TABLE NO. 15. GOLD & SILVER RECOVERIES IN SELECTIVE FLOTATION TESTS

Product	Test No. 4.			Test No. 10.			Test No. 12.		
	Assay Ounces Per Ton of Product								
	Wgt. %	Au	Ag	Wgt. %	Au	Ag	Wgt. %	Au	Ag
Heads		.10	1.32		.10	1.32		.10	1.32
Talc Froth	7.74	.08	2.16	8.26	.08	2.02	8.10	.08	1.84
Cu Conc.	8.08	.26	4.14	8.05	.22	3.88	8.28	.22	3.78
Zn "	9.47	.42	2.76	10.44	.46	3.62	12.20	.38	3.48
Fe Conc.	39.90	.06	1.10	36.35	.08	.90	27.80	.08	1.06
Tailings	34.80	.03	.53	36.90	Tr.	.42	43.60	Tr.	.40
	Recoveries Percentages								
Talc Froth		6.2	12.7		6.6	12.6		6.6	11.3
Cu Conc.		21.0	25.2		16.1	23.6		18.2	23.8
Zn "		39.8	19.8		47.0	28.6		36.4	32.2
Fe "		24.0	33.2		29.0	24.9		22.1	22.3
Tailings		10.4	14.0			11.5			13.2

TABLE NO.16. COMPARISON OF REAGENT AMOUNTS.

Reagents.	25-Ton Pilot Mill	Present Tests. Test No.10.	Difference
	Pounds Per Ton of Ore.		
Conditioning			
Lime	2.02	5.00	+2.98
Zinc Sulphate	2.00	2.67	+0.67
Talc Flotation			
Pine Oil	0.063	.24	+0.177
Copper Flotation			
Aerofloat or Pine Oil	0.09	.12	+0.03
NaCl	2.81	2.80	-0.01
Xanthate	.04	.04	-
Zinc Flotation			
Lime	2.55	2.00	-0.55
Copper Sulphate	.92	.75	-0.17
Cresylic Acid or Pine Oil	.15	.12	-0.03
Xanthate	.12	.08	-0.04
Total,	10.763	13.82	+3.057

TABLE NO. 17. METAL VALUES AND RECOVERIES.

Products	25-Ton Pilot Mill, Flin Flon, Man.						Present Investigation Selective Flotation - Test No. 10.					
	Analysis Per Cent (Au & Ag in Oz. Per Ton)											
	Wgt.	Fe	Cu	Zn	Au	Ag	Wgt.	Fe	Cu	Zn	Au	Ag
Heads			1.71	5.2	.056	0.90		24.74	1.90	5.09	0.10	1.32
Talc Froth	(Included in Tailings)						8.26	12.44	2.93	6.00	.08	2.02
Cu Conc.	8.68		14.28	4.1	.320	3.70	8.05	16.34	11.34	20.19	.26	3.88
Zn "	9.83		2.05	43.2	.055	1.78	10.44	17.34	5.35	25.01	.46	3.62
Fe "	(Included in Tailings)						36.35	43.12	.42	.53	.08	.90
Tailings	81.49		.33	0.74	.028	0.50	36.90	13.64	.17	.42	Tr.	.42
	Per Cent Recoveries or Losses											
Talc Froth	(Included in Tailings)								12.8		6.6	12.6
Cu Conc.			72.5	6.8	49.6	35.5			55.0		16.1	23.6
Zn "			11.8	81.6	9.7	19.4				88.0	47.0	28.6
Fe "	(Included in Tailings)							76.5			29.0	24.9
Tailings			15.7	11.6	40.7	45.1		19.4	3.5	3.1		11.5

CHAPTER 8.

BULK FLOTATION.

There are certain inherent difficulties in the way of effective bulk flotation for separation of all the sulphide minerals in an ore into one fraction from which the individual minerals may be selectively floated later. To obtain satisfactory results, it is essential that all the sulphide minerals be activated or promoted to maximum flotation conditions, and that there shall be a minimum of depressing effect or reagents present. When the selective flotation of the bulk flotation concentrate is made almost the reverse conditions are requited. That is, one only (at a time) of the sulphide minerals is to be promoted to flotation, the others are to be depressed.

In the tseatment of a copper-zinc-iron ore, such as the Flin Flon disseminated ore, bulk flotation will require an activator or promotor which will act equally effectively on all the sulphide minerals (chalcopyrite, zinc blende, pyrrhotite, and pyrite). The resulting floated concentrate has, then, to be treated by reagents in such a manner that the first, only the chalcopyrite will be left activated, or

made active to flotation whilst the other sulphide surfaces will be deactivated or depressed. Having selectively floated the copper, the balance of the bulk flotation concentrate has to be treated to reactivate the zinc blende without reactivating the iron sulphide minerals. The tailing product from zinc flotation would be an iron concentrate.

Obviously, there is another possible scheme of treatment of the bulk flotation concentrate, namely, to so treat or condition the bulk concentrate that the copper and zinc sulphide are depressed during the first retreatment, and an iron concentrate only floated; then to reactivate the zind blende only, leaving the copper minerals for a tailings from the selective treatment. There is, however, little likelihood of such a scheme working satisfactorily. In the first place, because of the relative floatability of the various sulphides, chalcopyrite is much more readily activated than it is depressed, and to keep it down for any length of time would require considerable quantities of the copper depressing agent. Also, the same substances which would activate the pyrite (if more were needed over that used in the bulk float) would also reactivate the copper. The zinc blende probably, could be handled either way. The substances used to depress the chalcopyrite may also depress the zinc blende, but the blende reactivation

would not be as pronounced. Probably the most important objection to this alternative procedure in selective flotation of this bulk float concentrate lies in the fact that pyrite is less readily floated than chalcopyrite while it is fairly readily depressed by the cheapest-of-all depressing reagents, namely, lime. It was, therefore, reasonable to eliminate this possible method of procedure without actual experiment and no investigations were made on it. The more obvious procedure of treating the bulk float for selective recovery of first the copper and then the zinc, leaving the iron as a selective flotation tailings, was adopted.

As has been mentioned earlier (Chapter 4.), talc may be handled in one of two ways in the bulk flotation, either included in the bulk floation concentrate and subsequently removed, or removed previous to bulk flotation. For the purpose of the present investigations it was considered advisable to remove it previous to bulk flotation; largely because this procedure eliminated another variable in the scheme of tests. Undoubtedly, to make the investigations complete, the alternative procedure should also be investigated.

Reagents for Bulk Flotation.

The results of the preliminary experiments on selective

flotation reported in Chapter 7, clearly indicated that there is a strong tendency for copper and zinc to float together, and no difficulty was anticipated in making the bulk float. But difficulties might arise in the subsequent selective treatment of that concentrate for separation of the copper and the zinc minerals.

C. S. Parsons (25) explains the tendency for zinc to accompany the copper in the selective flotation on the basis that there is produced in the Flin Flon ore, after grinding, a little soluble copper salt and this salt tends to promote the flotation of blende. Copper sulphate is the accepted reagent for reactivating zinc blende temporarily depressed for the previous selective float of another mineral; and hence the natural occurrence of a soluble copper salt can be accepted as the probable cause of the presence of zinc in the copper concentrate by selective flotation. It follows that little, if any, zinc activator would be required to raise all the zinc with the copper in the bulk float.

Because the iron sulphides carry a considerable portion, if not the major portion, of the precious metal values in the ore, it was necessary to float the iron, also, in the bulk float. The most common and cheapest depressant for pyrite is lime, and this would indicate that conditioning for the bulk

float should not include lime. Absence of lime in the circuit, however, would have the flotation circuit slightly acid, due to acidic decomposition products of the sulphides of the ore, and the selective flotation experiments had chearly indicated that an acid circuit was not satisfactory for talc removal. The alternative to lime appeared to be soda ash and thus preliminary experiments on bulk flotations required conditioning with soda ash for a neutral or alkaline circuit.

Procedure of Tests.

With the preceding theory in mind, the following scheme of procedure for bulk flotation tests was adopted,- Ore samples in unit batches of 1500 grams were to be conditioned in an alkaline pulp with soda ash under agitation for 30 minutes. Pine oil in amounts equivalent to 0.24 pound per ton of ore then added, and the talc froth produced, removed for 12 minutes. Following talc removal sulphide minerals were to be floated for 8 minute periods, using collecting and frothing reagents only. The remaining materials were to be collected as tailings. Bulk flotation concentrates from several batches were then to be treated under selective flotation procedures, similar to those used in the investigations on selective flotation of the raw ore itself,with preliminary conditioning follow-

ed by selective floats of first, copper and then zinc. The main problem appeared to be the determination of proper reagents and amounts for economical separation of the various valuable minerals from the bulk float concentrates.

C. S. Parsons (25) stated that copper salts (produced during grinding of the ore) activating the zinc blende, should be possible of elimination by sodium sulphide and suggested that selective flotation of bulk concentrate should be preceded by a conditioning with sodium sulphide and possibly a small amount of sodium cyanide. Selective flotation for copper also required the addition of zinc sulphate to depress the zinc, and of lime to depress the iron.

SULPHIDE MINERALS FLOTATION TESTS.

Preparation of sufficient bulk float concentrates to completely study the effects of variations in all these factors, required considerably more ore than was available. It was, therefore, necessary to limit the investigations to a study of a very few of the many factors. Actually, the investigations were limited to the determination of the effect of soda ash in the preliminary conditionining and the suitability of sodium sulphide as a deactivator for the zinc in copper flotation.

The results of the tests for sulphide minerals recoveries are shown in Table No. 18.

Discussion of Sulphide Minerals Flotation Results.

The results of the tests showed (Tests Nos. 1 and 2, Table No. 18.) that 0.133 pounds of thiocarbanilid or xanthate was not sufficient to completely recover the sulphide minerals. Test No. 2. showed that a depressant was required in the conditioning circuit in order that the copper and the zinc in the talc froth would be at a minimum.

Tests (Table No. 18.) showed that 4 pounds of lime and .200 pounds of xanthate yielded maximum copper and zinc recoveries, and low copper and zinc contents in the talc froth. Although, the lime depressed the pyrite, it was accepted as a standard conditioning reagent in preference to the soda ash.

COPPER FLOTATION TESTS.

The sulphide concentrates from the bulk flotation tests formed the feed to the copper flotation cells. The results of the tests are shown in Table No. 19.

Discussion of Copper Flotation Results.

Effect of Sodium Sulphide and Sodium Cyanide. The results of the tests (Table No. 19.) showed that increase of

sodium sulphide and sodium cyanide decreased the zinc in the copper concentrate. Increasing the sodium sulphide to 2.00 pounds and the cyanide to .02 pound showed no decrease of the copper recoveries; on the contrary, an increase was obtained. But increase of sodium sulphide above that amount decreased the recoveries.

ZINC FLOTATION TESTS.

The results of the zinc flotation tests are shown in Table No. 20.

Discussion of Zinc Flotation Results.

The zinc flotation results showed zinc recoveries up to 92.6 per cent. Increase of zinc depressing agents in the conditioning circuit, in order to decrease the zinc in the copper circuit, showed a slight decrease in the zinc recovery, but this was overcome by increasing the activator (copper sulphate) for zinc blende. Decreasing the zinc in the copper float increased the zinc recoveries.

Recovery of Gold and Silver.

The gold and silver concentrations (Table No. 21.) in the flotation products were analogous to those of the straight selective flotation process. Appreciable amounts of the gold (26.1%) and the silver (24.5%) remained in the tail-

ing.

SUMMARY AND CONCLUSIONS.

The results of the bulk flotation tests may be summarized as follows,-

1. The bulk flotation followed by a selective flotation process, for the separation and concentration of the Flin Flon ore, was possible.
2. A depressant for the sulphide minerals was necessary in the conditioning operations in order to decrease the metallic minerals in the talc froth. Lime served this purpose fairly well.
3. Four pounds of lime and .200 pound of xanthate yielded maximum copper and zinc recoveries.
4. Sodium sulphide and sodium cyanide decreased the zinc in the copper concentrates.
5. Over 65 per cent of the gold and over 75 per cent of the silver floated with the sulphide minerals concentrates.
6. The reagents and amounts which yielded most satisfactory recoveries in the bulk flotation were,-

To conditioning Circuit:-		
Lime	4.00	pounds
To Talc Cell:-		
Pine Oil	.24	"
To Sulphide Minerals Cell:-		
Xanthate	.20	"
Pine Oil	.24	"

These reagents gave recoveries of the copper, 94.1 per cent of the zinc and 60.8 per cent of the iron in the bulk float.

7. The most satisfactory recoveries and the grades of concentrates in the selective flotation of bulk float were obtained with the following reagents and amounts,-

To Conditioning Circuit:-		
Lime	3.00	pounds
Sodium Sulphide	2.00	"
Sodium Cyanide	.20	"
Zinc Sulphate	4.00	"
To Copper Cell:-		
NaCl	2.80	"
Xanthate	.02	"
Pine Oil	.06	"
To Zinc Cell:-		
Lime	1.00	"
Xanthate	.04	"
Cresol	.06	"

These reagents gave recoveries of 55.7 per cent of the copper and 84.3 per cent of the zinc fed to the selective flotation circuit.

COMPARISON WITH SELECTIVE FLOTATION.

A comparison between the reagent consumptions, grade of

concentrates, and recoveries of metalsin the bulk flotation with subsequent selective flotation and the straight selective flotation tests is shown in Tables Nos. 22 and 23.

Reagent Consumptions.

The reagents consumption is appreciably higher. This is due to the fact that the minerals had been activated in the bulk flotation operation, and greater amounts of depressing reagents were required, in the subsequent operations, to overcome this effect.

Recoveries.

The recoveries of the copper and the zinc were practically the same for the two methods. The iron recovery was not as high in the bulk flotation.

The bulk flotation process yielded a slightly higher grade of copper concentrate, and appreciably higher zinc concentrate. (Table No. 23.).

The gold and silver concentrations in the flotation products were analogous. The bulk flotation tailings carried higher precious metal values. This was probably due to lower recovery of the iron minerals. No doubt, the values would have been the same had the iron recoveries been as high.

FLOW SHEET.

The proposed flow sheet is shown in Figure No. 17.

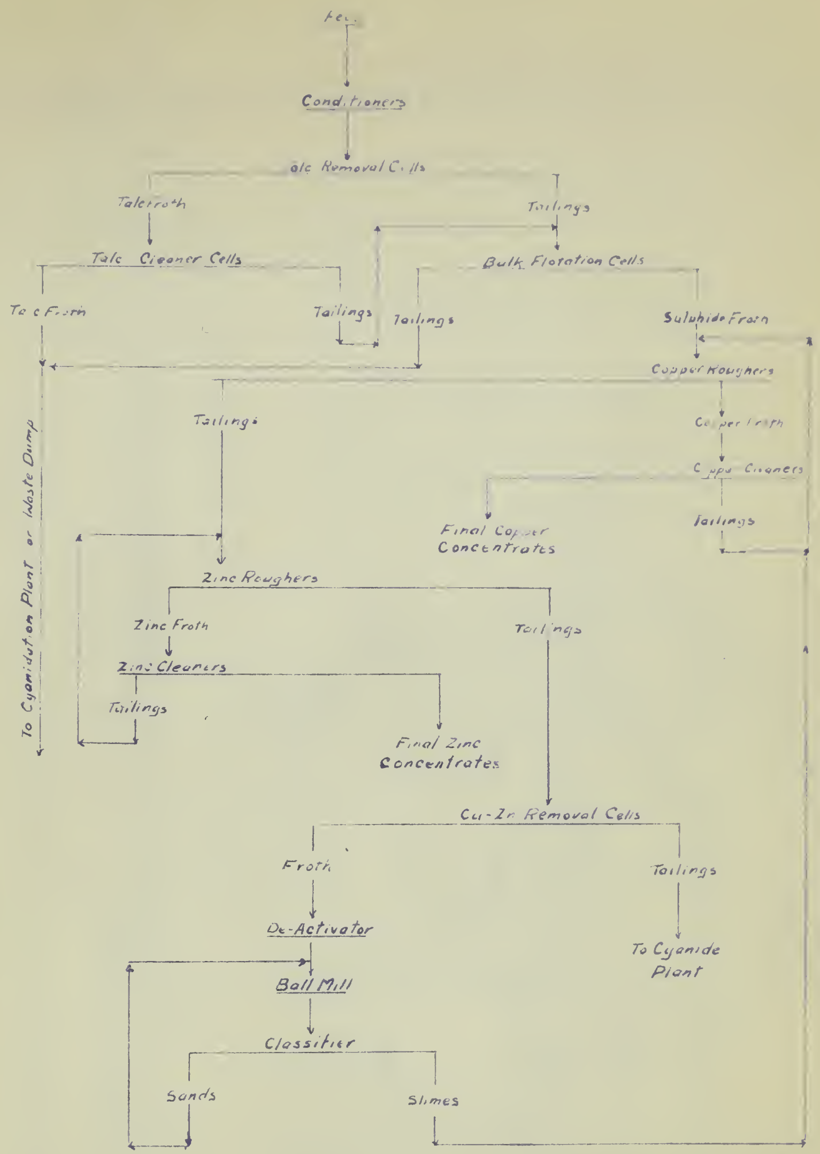


FIG. 17 PROPOSED FLOW SHEET FOR BULK FLOTATION

TABLE NO. 18. BULK FLOTATION TESTS SULPHIDE MINERAL RECOVERIES.

Test No.	Reagents Pounds Per Ton of Ore: Conditioning CaO	Conditioning Na_2CO_3	Talc Zinc Oil	Sulphides KEtX	Sulphides Thio	Sulphides Pine Oil	Products Name	Wgt.	Percentages Analysis Fe	Analysis Cu	Analysis Zn	Recoveries or Losses Fe	Recoveries or Losses Cu	Recoveries or Losses Zn
							Heads		24.74	1.90	5.09			
1	5.33		.24		.133	.24	Talc Froth	12.66	Unsuccessful, sulphide minerals were not promoted sufficiently.					
							Sulphide Con	18.86						
							Tailings	77.03						
2	5.33		.24	.133		.24	Talc Froth	7.35						
							Sulphide Con	16.17						
							Tailings	76.50	Unsuccessful, sulphide minerals were not sufficiently promoted.					
3		2.67	.24	.267		.24	Talc Froth	13.85	14.2	3.73	8.64	8.0	27.2	23.5
							Sulphide Con	49.55	40.2	2.50	6.35	87.5	89.5	80.7
							Tailings	36.60	11.4	.26	.94	11.1	7.7	14.8
4	3.00		"	"		"	Talc Froth	9.44	12.1	2.96	5.95	4.1	14.7	11.0
							Sulphide Con	49.24	36.4	2.92	8.29	79.1	89.0	90.2
							Tailings	41.20	15.7	.34	1.24	19.7	9.5	8.9
5	4.00		"	.200		"	Talc Froth	8.12	11.4	2.70	4.94	3.8	11.6	8.0
							Sulphide Con	9.61	36.5	3.85	11.09	60.8	90.2	94.1
							Tailings	52.20	18.9	.25	.61	37.7	8.8	5.5
6	4.00		"	"		"	Talc Froth	8.05	12.2	2.35	4.96	4.0	10.0	7.9
							Sulphide Con	40.35	33.5	3.81	10.60	57.0	90.3	91.3
							Tailings	51.60	19.1	.24	.56	41.2	8.8	8.1

TABLE NO.19. SELECTIVE FLOTATION TESTS OF THE BULK FLOTATION CONCENTRATES FOR THE RECOVERY OF COPPER.

Test No.	Reagents Pounds Per Ton of Ore							pH	Products - Percentages					
	Conditioning				Copper						Analysis			Recovery
	CaO	Na_2S	NaCN	$ZnSO_4 \cdot 7H_2O$	NaCl	KEtX	Pine Oil		Name	Wgt.	Fe	Cu	Zn	Cu
									Heads		40.2	2.50	7.15	
3	2.67	.67	.007	2.00	2.80	.02	.06	11.0	Cu Conc.	12.9	15.8	8.78	34.86	45.2
									Heads		36.4	2.92	8.29	
4	2.67	1.00	.010	3.00	2.80	.02	.06	11.5	Cu Conc.	13.1	15.8	10.08	34.08	48.5
									Heads		36.5	2.70	11.09	
5	3.00	2.00	.020	4.00	2.80	.02	.06	11.9	Cu Conc.	12.0	16.6	12.51	27.05	55.7
									Heads		33.5	3.81	10.60	
6	3.00	3.00	.020	4.00	2.80	.02	.06	12	Cu Conc.	5.6	21.2	14.44	15.74	21.3

TABLE NO.20. SELECTIVE FLOTATION TESTS OF THE BULK FLOTATION CONCENTRATE FOR ZINC RECOVERY.

Test No.	Reagents - Pounds Per Ton of Ore									pH	Products - Percentages					
	Conditioning				Zinc Flotation Cells						Name	Wgt.	Analysis			Recovery
	CaO	Na_2S	NaCN	$ZnSO_4$ $7H_2O$	CaO	$CuSO_4$ $5H_2O$	KEtX	Pine Oil	Cresol		c		Fe	Cu	Zn	Zn
3	2.67	.67	.007	2.00	1.00	0.67	.04	.06			Cu Conc.	12.9	15.8	8.78	34.86	
										11.6	Zn "	14.5	49.9	6.85	14.24	77.5
4	2.67	1.00	.010	3.00	1.00	1.00	.04		.06		Cu Conc.	13.1	15.8	10.08	34.08	
										11.5	Zn "	13.5	24.6	6.34	22.79	81.0
5	3.00	2.00	.020	4.00	1.00	1.00	.04		.06		Cu Conc.	12.0	16.6	12.51	27.05	
										11.8	Zn "	17.3	15.5	6.62	38.34	84.3
6	3.00	3.00	.020	4.00							Cu Conc.	5.6	21.2	14.44	15.74	
										11.9	Zn "	29.7	18.6	7.56	30.32	92.6

TABLE NO. 21. RECOVERY OF GOLD AND SILVER. TEST NO.5.

Product	Wgt.%	Assay oz. Per Ton of Product		Recoveries Percentages	
		Au	Ag	Au	Ag
Heads		.10	1.32		
Talc Froth	8.12	.07	2.00	5.7	12.3
Cu Conc.	4.72	.26	3.88	12.4	14.0
Zn Conc.	6.84	.40	4.50	27.3	23.3
Fe Conc.	28.05	.10	1.63	28.1	34.7
Tailings	52.20	.05	.61	26.1	24.5

TABLE NO. 22. COMPARISON OF REAGENT CONSUMPTION.

Reagents	Selective Flotation Test No.10.	Bulk Flot. Test No.5.	Difference
	Pounds Per Ton of Ore		
Conditioning Bulk Flot.			
Lime		4.00	+4.00
Talc Flotation			
Pine Oil	.24	.24	
Sulphide Flotation			
Xanthate		.20	+ .20
Pine Oil		.24	+ .24
Conditioning			
Lime	5.00	3.00	-2.00
Zinc Sulphate	2.67	4.00	+1.33
Sodium Sulphide		2.00	+2.00
" Cyanide		.20	+0.20
Copper Flotation			
NaCl	2.80	2.80	
Xanthate	.04	.04	- .02
Pine Oil	.12	.06	- .06
Zinc Flotation			
Lime	2.00	1.00	-1.00
Copper Sulphate	.75	1.00	+ .25
Xanthate	.08	.04	- .04
Cresol	.12	.06	- .06
Iron Flotation			
Xanthate	.20		- .20
Pine Oil	.36		- .36
Total,	14.38	18.86	+4.48

TABLE NO. 23. METAL VALUES AND RECOVERIES.

Product	Selective Flotation Test No. 10.						Bulk Flotation Test No. 5.					
	Analysis Per Cent (Au & Ag in Oz. Per Ton)											
	Wgt.	Fe	Cu	Zn	Au	Ag	Wgt.	Fe	Cu	Zn	Au	Ag
Heads		24.74	1.90	5.09	.10	1.32		24.74	1.90	5.09	.10	1.32
Talc Froth	8.26	12.44	2.93	6.00	.08	2.02	8.12	11.4	2.70	4.94	.07	2.00
Cu Conc.	8.05	16.34	11.34	20.19	.26	3.88	4.72	16.6	12.51	27.05	.26	3.88
Zn "	10.44	17.34	5.35	25.01	.46	3.62	6.84	15.5	6.62	38.34	.40	4.50
Fe "	36.35	43.12	.42	.53	.08	.90	28.05	40.5	1.62	1.58	.10	1.63
Tailings	36.90	13.64	.17	.42	Tr.	.42	52.20	19.1	.24	.56	.05	.61
	Per Cent Recoveries or Losses.											
Talc Froth					6.6	12.6					5.7	12.3
Cu Conc.			55.0		16.1	23.6			55.7		12.4	14.0
Zn "				88.0	47.0	28.6				84.3	27.3	23.3
Fe "		76.5			29.0	24.9		60.8			28.1	34.7
Tailings		19.4	3.5	3.1		11.5		39.2	6.8	6.3	26.1	24.5

CHAPTER 9.

CONCLUSIONS.

Conclusion.

The experimental work showed that a preliminary bulk flotation followed by a selective flotation method would yield, if not higher recoveries and grade of concentrates, at least as high as a straight selective flotation. With higher reagents consumption to attain the results equally as high, no advantage could be gained by adopting this practice for the separation and concentration of the Flin Flon ore.

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